THE PRACTICE

OF

ORE DRESSING

IN EUROPE.

A DESCRIPTION OF FOREIGN METHODS FOR THE MECHANICAL CONCENTRATION OF ORES, REVISED AND CORRECTED FROM A SERIES OF ARTICLES IN THE SCHOOL OF MINES QUARTERLY OF COLUMBIA COLLEGE.

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PREFACE.

A VISIT to the ore dressing establishments of Europe and a comparison of their several methods of working discloses the great development which the mechanical treatment of ore has undergone during the past few years. Some of the works have remained almost unaltered since the date of their erection, twenty years ago; others have been enlarged or modified in accordance with advanced ideas, and sometimes exhibit the old and the new side by side; and others, again, have but recently been erected and embody many of the improvements derived from experience. To note the advances that have been made in ore dressing may prove of interest to some, and would certainly possess practical value if suggesting the direction in which further improvements will be sought. Most conducive to this object would be a study of the schemes of ore treatment that have been developed in the various mills; but this method, unless thoroughly carried out—a proceeding which would far exceed the limits here proposed—can lead to no useful results: any scheme or synopsis of an ore treatment not supplemented by an accurate description of the ore, and by accounts of the disposition made of the various mill products, and by a complete statement of local conditions, including many small, yet influencing details, will always prove very unsatisfactory. Remarks of a general nature have therefore been preferred, and these (restricted so as not to embrace the subject of coal washing) present a digest of matter that has been gathered in the foreign mills through the kindly courtesy of their owners and managers.

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ORE DRESSING IN EUROPE.

GENERAL PRINCIPLES.

MOST ore masses require crushing to unlock or set free their constituent minerals. The richer pieces of crushed material may bear the cost of manual treatment, by which the precious "mineral" is completely separated from barren "gangue," but the great bulk of ore must always be treated by less costly, though more complicated, mechanical means, in order to separate its different constituents. Such separations can be effected in most ores of ordinary quality, when they are reduced to pieces of small size, by the successive operations of *sizing* and *sorting*. Sorting is usually a separation of equal-falling grains from those moving with a different velocity in the rising current of a fluid medium. When the fluid is air, the dressing process is said to be *dry;* when water, the process is *wet;* and it is the wet process which is used in the great majority of cases. The falling velocity of each ore grain depends under these circumstances upon its size and its specific gravity. If the ore be classified by screening into a number of sizes, with a view that the smallest grain of the heavier mineral shall fall faster than the largest grain of the lighter mineral, then in the subsequent sorting of each class of such sized material the specific gravities of the grains of different minerals will determine their separation. When, however, sorting precedes sizing, then the equal-falling sorted grains will be of different sizes—the specifically lightest mineral being in the condition of the largest particles, and consequently a sizing of such sorted material must result in a separation of the different minerals.

The sorting apparatus most commonly used for all the coarser ore sizes is a *jig*, which consists essentially of a sieve set into a box and immersed in the fluid separating medium. Ore is charged upon the sieve and is subjected to quick alternations of short rises and falls, due either to an up-and-down

motion of the sieve, or imparted by the flux and reflux of currents of water or air forced by a reciprocating piston or a bellows through the sieve when the latter is fixed. The effect of this "jigging" is to bed the material in layers of different specific gravities. Ores of small *nut*, *pea*, and *sand* sizes,[1] from $1\frac{1}{8}$ to $\frac{1}{25}$ of an inch in diameter, are treated first by sizing and then by jig sorting.

The order of operations is usually reversed in treating ore that is less than $\frac{1}{20}$ or $\frac{1}{25}$ inch in size. As a rule such fine material is first sorted by a continuous upward, or a nearly horizontal, current of water or air; the heavier particles, both large and small, fall through the current, while the lighter ones are seized and borne off by it. Following upon this classification may come a separation by jigging, which in this case produces an effect of sizing (as hereafter explained), or the fine ore suspended in water and known as *slime* is subjected to a species of sizing on inclined surfaces by which the larger particles—those that roll the better or that are easily acted upon by a thin sheet of flowing water, become separated from the smaller (and now, after sorting, the specifically heavier) particles which adhere closely to the surface of the washing apparatus. The separation of the different minerals of fine, sorted ore by this wet sizing has received the name of "slime washing." An analysis of the action of this washing is always compli-

[1] The terms *nut, pea, sand, meal* and *pulp* are used to designate definite sizes as given in the following table, which is reproduced from the *School of Mines Quarterly*, Vol. II., p. 216.

MILLIMETRES.	INCHES.	INCHES.	WIRE CLOTH.	NO. WIRE.	NAME.	
64.	2.56	$2\frac{1}{2}$	3 in. mesh,	00	Large	Nut.
45.2	1.80	$1\frac{3}{4}$	$2\frac{1}{8}$ " "	0		
32.	1.28	$1\frac{1}{4}$	$1\frac{1}{2}$ " "	2	Small	
22.6	.90	$\frac{7}{8}$	$1\frac{1}{8}$ " "	4		
16.	.64	$\frac{5}{8}$	$\frac{13}{16}$ " "	7	Large	Pea.
11.3	.45	$\frac{7}{16}$	$\frac{5}{8}$ " "	8		
8.	.32	$\frac{5}{16}$	No. $2\frac{1}{2}$ "	12	Small	
5.6	.22	$\frac{1}{4}$	" $3\frac{1}{2}$ "	14		
4.	.16	$\frac{3}{16}$	" $4\frac{1}{2}$ "	15	Coarse	Sand.
2.8	.11	$\frac{1}{8}$	" 6 "	16		
2.	.08	$\frac{3}{32}$	" 8 "	18	Fine	
1.4	.055	$\frac{1}{16}$	" 12 "	22		
1.	.04	$\frac{1}{25}$	" 16 "	25	Coarse	Meal.
0.71	028	$\frac{1}{32}$	" 24 "	28		
0.5	.02	$\frac{1}{50}$	" 35 "	31	Fine	
0.35	.014	$\frac{1}{64}$	" 50 "	34		
0.25	.01	$\frac{1}{100}$	" 70 "	38	Pulp.	

cated; the ore particles are acted upon by a variety of forces, which are not under complete control, and the efficiency of the process cannot be considered altogether satisfactory.

In reviewing foreign dressing practice, the order adopted is that naturally developed in following the ore from its crude condition in the mine to its final form as a marketable concentrate.

UNDERGROUND SEPARATION.

A manual separation of ore from easily distinguishable barren rock is always undertaken in the mines, and it is not uncommon practice to select, underground, pure galena ore and to hoist it separately so as to minimize losses of "mineral," due to attrition and handling. The miner receives a slight premium for this rich soft ore, but not sufficient to tempt him to devote any of his time to spalling or true dressing of the material.

Sizing is seldom performed underground, yet it is sometimes carried out, and notably in very deep mines, such as are worked, for example, at Przibram,[1] in Bohemia. The ore is there hoisted from many levels; but the mode of extraction that has been adopted—the system of counter-balancing the ascending cage by the descending one—does not permit of hoisting first from one level and then changing at pleasure to any other; the ore must therefore be stored on the different levels while awaiting its turn to be hoisted. Masonry bins near the hoisting-shaft are used for this purpose; each bin is covered with an inclined grate or "grizzly" upon which the ore is dumped. Two ore classes of different sizes are produced by the operation and are accumulated in separate compartments of the bin. In this way the first sizing is effected without involving any extra expense in labor or handling.

GENERAL SIZE CLASSIFICATION.

The first general sizing, which can rarely be performed to advantage underground in the way above described, is more commonly effected as the ore comes from the mines and is tipped upon grates with 64 to 70 mm. ($2\frac{5}{8}$ inches)—and less fre-

[1] Description of Ore Dressing at Przibram, Transactions of American Institute of Mining Engineers, 1881, vol. ix., p. 420.

quently 90 mm.—openings. What falls through the grates, the *mine stuff*, *mine finings*, or *fine mine ore*, is advanced at once to further treatment; the *lump ore*, or that portion which passes over the grates, in pieces rarely larger than *head* size, might be broken up and then be made to rejoin the fine ore and pursue with it a common course. An advantage, however, is frequently found in keeping these two classes of ore separated to the very end, for their characters may present marked differences. The fine mine ore is that which has broken into small pieces in mining and subsequent handling; it is apt to be more friable and richer than the lump ore, and to carry much of the coarsely divided mineral. It contains but few coarse pieces of totally barren rock, and its gangue is largely composed of vein matter. The lump ore, on the other hand, is frequently the tougher; it includes, as a rule, most of the "country rock" that is mixed with the ore proper, and generally carries, after crushing, much barren rock of *nut* sizes. It is sometimes noticeable that each class of material shows the preponderance of a different set of the constituent minerals in the ore. These points of difference are sufficient to influence all the dressing operations. The hand-picking and cobbing will no longer be just the same for the two classes, and there will be an evident advantage in not mixing them, for this would only complicate the work of manual sorting for such simple intelligence as that of the children who are engaged at it. The jigging processes must also differ. The jigs for the crushed lump ore will probably be larger than those for the fine mine ore because of the considerable quantity of coarse barren rock contained in the first named class of material,[1] and the points at which jig tailings can be discarded are not the same in both cases. Finally, the slime treatments may be radically different, as when, for example, a granular quartzose gangue prevails in one class and a slaty gangue in the other.

The proportion of finings to the whole output varies between wide limits, even in one mine. The amount increases with the use of high explosives, and may frequently be found to range between 50 and 65 per cent. of all the ore.

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[1] The dressing works at Ems, in Nassau, illustrate this fact. Description of Ems mills, Berg und Huettenm. Zeitung, 1882, vol. xli., pp. 289 *et seq.*

CLEANSING.

The grit, sand and slime which coat the ore, and more especially the fine mine ore, as it is brought to the surface, are washed off to facilitate manual sorting and keep the work of jigging neat. The mine finings are usually cleansed while being classified into several sizes: a sharp spray of water either plays upon and passes through the sizing screens, or, to be more effectual when revolving screens are employed, a fixed hollow pipe is used as axle of the screen, and the water issues from it directly upon the ore.[1] Warm water from a steam condenser is of considerable advantage in the cold season, because of the hand-picking which follows sizing. The dirt derived from the washing is collected in vats and treated with other unclassified sands. The cleansing of the lump ore intended for spalling, or coarse hand dressing, can almost always be effected by merely playing a hose upon it; but lump ore that is to be crushed by machines is not subjected to cleansing, nor is washing found necessary after the crushing of this ore and preparatory to hand-picking.

Certain classes of ore, however, and notably zinc ores and some ores of iron, contain a quantity of stiff clayey gangue which frequently envelopes the ore pieces, large and small, and cements them into masses of considerable size. Simple washing will not remove the clay; it becomes necessary to pass all the ore through a special cleansing apparatus—for example, through a large revolving conical drum, some fifteen to eighteen feet long, and twelve feet in diameter, in which the ore, besides being exposed to the vigorous action of water, is raised by upturned blades, and then falling upon other blade edges gradually breaks loose from the clay and is cleansed.[2] At the discharge end of the washer there is a draining screen; the cleansed ore passes over it and then generally drops upon the sizing grates which separate the lump from the finings.

SPALLING.

The practice of spalling, or breaking up and sorting the

[1] This method is applied at the Lautenthal dressing works in the Upper Harz region, Prussia.

[2] A new mill at Blei Scharley, near Beuthen, Upper Silesia, running on oxidized and sulphuret zinc ores with a clayey and dolomitic gangue, carries out this kind of cleansing very thoroughly.

lump ore by hand and sledge, is falling more and more into disuse. The impoverishment of the mineral deposits in some of the old mining regions has necessitated the treatment of larger quantities of ore than heretofore, while this increased demand has diverted much of the labor of the district from surface work about the mills to operations underground. As a consequence of these conditions, as well as owing to the generally increased value of labor everywhere, compared with twenty years ago, the lump ore is now crushed by machines before being subjected to any kind of manual sorting, and spalling is becoming restricted to the comparatively few lumps of rich ore which can be selected in the mine or are more commonly picked off the grate upon which the ore is first sized. The rich ore is dressed by hand to avoid subsequent losses from partial pulverization.

There is, however, an important exceptional case in which spalling still holds its own in the modern mill. When, for example, an ore carries argentiferous galena and zinc blende with copper and iron pyrites, spathic iron and quartz, the result of crushing would be to break up much of the blende, pyrites, and iron into grains, which, with specific gravities ranging only from 3.9 to 5.1, could not be practically separated from one another by jigging or any ordinary mechanical dressing process; a mixture of these minerals has no market, and hence crushing here would obviously result in loss. The ore at Ems furnishes a good illustration; it contains all of the above named minerals, and is subjected to spalling. The products of this operation are: 1, *Rich galena ore;* 2, *cobbing ore;* 3, *pyritiferous cobbing ore;* 4, *barren rock.* Cobbing is in this case but a further development of spalling, and by considering it now the explanation of the use of spalling may become more apparent. Class 1, rich galena ore, is treated by careful work-boys (who guard against producing much dust), and yields pure galena and further by-products for mechanical dressing. Class 2, cobbing ore, is divided by cobbing into nine sub-classes, as follows:

a. Very rich galena ore, treated with Class 1.

b. Rich galena and blende; sent to special treatment in blende dresser.

c. Marketable blende; sold.

d. Moderately rich galena ore, with gangue of quartz, accumulated and treated separately at the mill.

e. Moderately rich galena ore, with gangue of spathic iron.[1]

f. Moderately rich blende ore with gangue of quartz.[1]

g. Moderately rich blende ore with gangue of spathic iron; discarded.

h. Poor ore; sent to crushing rolls.

i. Marketable spathic iron ore; sold.

j. Barren rock; discarded.

Class 3, pyritiferous cobbing ore, is similarly treated, and yields:

k. Marketable copper pyrites, first quality; sold.

l. Marketable copper pyrites, second quality; sold.

m. Pyritiferous ore mixed with galena.

n. Pyritiferous ore free from galena; accumulated and concentrated by dressing, or sold.

o. Barren rock; discarded.

The products *d*, *e*, *f*, *m*, and *n* can be treated separately without any difficulty, and as the only way by which it seems possible to class the lump ore into these groups without incurring great losses of mineral is by a process of cobbing preceded by spalling, the continuance of this last named operation appears justified. In many localities, however, the value of labor is too high to permit of its application, and at best it is a painfully primitive practice which, when carried out on a large scale, retards, or even prevents, by the sedentary nature of its work, a healthy physical development of the youth of the mining population. It is to be hoped, therefore, that eventually other properties in minerals, beside their specific gravities, will be put into common practice when such separations as those above mentioned are to be effected, and that then the necessity for spalling will disappear.

Rock Breaking.

The Blake, or the Marsden-Blake, form of jaw crusher is widely applied for rock breaking. The size to which lump ore is broken is commonly 2½ to 2¾ inches—a size convenient for hand-picking. That there is a great advantage in setting up the crushers in a building apart from the main mill, or separated from it by a substantial partition, has been repeatedly demonstrated. Crushing produces some very fine and often very sharp dust, and this, if allowed to spread and settle in the

[1] Accumulated and treated separately in the mill.

mill, hastens the wear of the dressing machinery in a very marked degree. Sometimes, when other dispositions have not been practicable, the sizing operation which always follows upon coarse crushing is performed on screens which are placed in the same apartment with, and directly beneath, the crushers, while further treatment is carried out in a separate building.

The coarsely crushed ore and the mine finings begin parallel courses of treatment by being classed into several sizes; the operations which ordinarily succeed this sizing are hand-picking and cobbing for a portion of the material, and for another portion mechanical sorting on jigs; then some of the products of the manual sorting and the jigging are crushed by rolls; after this there is further sizing and thereupon fine jigging of the roll-crushed ore, followed sometimes by repicking; then finer crushing of the ore which remains, and so on, repeating the cycle of operations—reduction, sizing, and sorting—until all remaining mineralized material is in the condition of sand and slime, from which the fine mineral is obtained by repeated washings. The whole mode of economical dressing is based on the oft enunciated principle that, to obtain the "mineral" with a minimum loss from comminution, the ore must be broken up only just enough to unlock all the mineral it may contain down to a certain size, and that this freed mineral (and also freed barren rock) must be separated from the mass *at once;* the ore which then remains requires finer crushing to unlock more finely divided mineral, and this, in its turn, must be separated as before. The number of operations that may be necessary, as well as the number of machines required for each one, can only be known through an intimate acquaintance with the ore. Preparatory experiments on a small scale aid very largely in revealing the nature of the material that is to be dressed, but yet in a new mill, beginning to work on ore that has never been treated in a large way, considerable room is allowed for supplying deficiencies or making alterations, as experience may suggest.

SIZING.

The ore that is delivered upon the sizing screens ranges from the condition of fine sand to that of pieces 64 or 70 mm. ($2\frac{3}{4}$ inches) in diameter. Whenever the ore contains coarsely divided mineral, which is with few exceptions the common case, one size-class is likely to be formed of all the material

coarser than 30 or 32 mm. (about 1¼ inch); more rarely 50 or 55 mm. (2 inches) is the inferior limit. This class, known as *picking ore*, is subjected to manual treatment. The material ranging from 32 mm. (or possibly 55 mm.) down to 3 or 4 mm. (approximately from 1¼ to ⅛ inch diameter) is separated into several sizes—as many, perhaps, as six or seven, or as few as only two. These classes are treated by wet jigging, and hence their name *jigging ores*.

The practice just alluded to, in which only two size-classes are made between 32 mm. and 3 mm., is remarkable and quite unusual. It is found in Cornwall in the dressing of copper pyrites with a rock gangue, in a case where concentrates of very poor grade answer the commercial requirements; it also occurs in dressing works at Lauremburg,[1] in Nassau, in the treatment of an ore of argentiferous galena and blende with a siliceous gangue. The very satisfactory results there obtained in the jigging which follows immediately after this sizing (and in which the system of sorting for intermediate products is developed), bears proof that so small a number of size-classes is sufficient for that particular ore.

The coarsest class of jigging ore—material ranging down in a few cases as low as 15 or 18 mm. (⅝ inch)—was formerly subjected in some works, for example in those at Clausthal[2] in the Upper Harz, Prussia, to small or *fine picking*, as it is called, but now all works, without exception, follow the practice of treating that ore by hydraulic sorting, and they restrict the fine picking, if any, to the head products or concentrates of the coarsest jigs.

The *fine sand* and *meal*—all that passes through screens of 3 or 5 mm. (approximately No. 6 to No. 3½ mesh) are treated at certain mills in a species of hydraulic classifier, to produce several grades of material for fine jigging, and for slime washing. By this practice, one kind of hydraulic sorting, namely jigging, follows immediately after another upon the same lot of ore—a system of dressing which, as hereafter explained, has a *raison d'être* but is not to be recommended for the generality of ores, so long as the degree of fineness does not prevent efficient sizing. The better plan, therefore, and the one generally ap-

[1] Description of the Lauremburg Dressing Works, Berg u. Huettenm. Zeit'g, 1882, vol. xli., pp. 140–144.

[2] Description of Clausthal Dressing Works, *Ibid.*, 1882, vol. xli., pp. 29 *et seq.*; also Transactions Am. Inst. of Mining Engineers, 1877, vol. vii., pp. 470 *et seq.*

plied, is to size this fine stuff, producing three or four classes of *fine jigging ores*, larger than 1 mm. ($\frac{1}{25}$ inch), and to treat all material under 1 mm. or under 1½ mm. in the slime department. In parts of Austria sieve sizing is carried down to ½ mm., but the limits for satisfactory work have been set down by an engineer[1] of long experience at 1.4 mm.

The proportional quantities of the several sized ore classes must naturally vary greatly in different mills. Mine finings are often found to yield between 40 and 50 per cent. of picking ore, 45 to 55 per cent. of jigging ores, and 2 to 4 per cent. of slime. Tough quartzose rock yields, after coarse crushing and sizing, about 70 to 80 per cent. of picking ore, and the remainder as jigging and fine jigging ores, with a small percentage of slime.

The custom of determining the screen mesh sizes down to the hundredth of a millimetre has been abandoned; theoretical formulæ are still applied, but the coefficient of experiment—the result of tests with hand sieves and small hand jigs—now bears an important part in fixing upon the sizes. At a few dressing works the factor of the geometrical proportion in which the mesh sizes increase is as low as 1.35; in others it is as high as 2, while again in many others there is no fixed factor at all. The evident tendency is to simplify the works by ceasing to carry the sizing classification to the extreme which it may be said to have reached at several prominent mills. A little extra work is thereby thrown upon the jigging, because now, more than ever, mixed products are formed in this operation. To care for these products all the jigs—even to the coarsest—have at least three sieves, of which the second one, smaller sometimes than the rest, collects the "middlings." The limit to reduction in the number of size classes is reached when barren tailings, or such as carry only occluded mineral, can no longer be produced in jigging.

Either shaking riddles or revolving sieves, known as trommels, can be used for sizing. The former undoubtedly perform the cleanest and best separations: being run at 150 to 200 sharp shocks per minute, and with the whole screening surface available, it is natural that one of these should have far higher capacity and efficiency than the trommel, in which the effective screening surface is but a narrow strip, 8 to 12 inches

[1] Mr. Heberle, constructor of ore dressing machinery at the Humboldt Machine Shops of Kalk, near Cologne.

wide, while the number of shocks it imparts to the ore—shocks that are gentle in nature, and produced through frictional adhesion of the ore to the side of the trommel, in that this carries the stuff up a short distance to let it return, sliding and dropping to the bottom—hardly reaches to double the number of sieve revolutions, or not over 40 to 60 impulses per minute. But if riddles size well they also require much motive power, and impart deleterious shocks to the structure of the mill, and, as a serious drawback, they wear very rapidly and necessitate frequent stoppages for repairs. The weight of these disadvantages, strengthened by the consideration that trommels of ordinary dimensions can, as a rule, easily meet all requirements as regards capacity, have, rightly or wrongly, led to the complete abandonment of riddles for heavy work in ore dressing.[1] They have been thrown out in favor of revolving sieves at the Clausthal mills and at a number of other dressing works. Small riddles, suited for light work, are, however, frequently employed. When, for example, a rock breaker is used—and perhaps not continually—to crush a moderate quantity of ore, the subsequent sizing into three or four classes can readily be effected by a simple zigzag shaking riddle actuated by the pitman or the jaw of the crusher—an arrangement which costs much less than a set of trommels for the same work. Light shaking riddles are suspended in some mills under crushing rolls to effect a rough classification of material, which then passes to drum-screens for further sizing.

The revolving sieves are either conical or cylindrical. Coned trommels are used at present about as much as cylindrical ones, and may in time become generally preferred to the earlier form. One old objection to the conical trommel no longer exists—manufacturers now supply promptly and at reasonable price sieves which suit the frames of coned drums. Conical trommels on horizontal axes are more easily mounted than the cylindrical ones, which must necessarily be set with axes inclined. In some modes of setting, when the drums are placed in steps side by side, those of conical form can be arranged with greater economy as regards height; they are always the easier kind to manage and require the least motive power.

Each sizing drum frequently contains two or three fields of

[1] Riddles are still much used in coal sizing, wear being in that case less rapid and great capacity a more important factor.

different meshes. The coarsest ore sizes are, almost without exception, separated from the whole mass of material in the first trommel, while the finest sizes are classified in the last one, but in each separate trommel the sizing commonly takes place in inverse order, the finest field being at the head of the trommel, and the coarsest one at the discharging end. Such a disposition in each trommel naturally hastens the wear of the finer sieves, but these are easily patched without incurring sensible delay and are made to last until the whole trommel needs repairs. The objection to the plain trommel on this ground does not, therefore, seem to be considered a serious one by most of the dressing works, and it continues in general use, to the exclusion of the more expensive step-trommel and other devices which separate and discharge the coarsest ore size first. Nevertheless, prominent engineers, recognizing other disadvantages in the plain trommels with several fields, have repeatedly endeavored to introduce some of the newer forms in cases where limited space or height did not admit of using single field trommels placed one beneath the other in steps.

In the plain cylindrical trommel of several fields, as usually employed, the ore falls at the charging end upon the finest sieve of the drum. Disregarding for a moment the extra wear of this fine sieve, this method yet remains a poor one, for a fine separation is carried out while several coarse size classes are still contained in the ore, tending to block the passage of the fine stuff, and so interfering with good work. When the trommels are conical and the ore enters, as in common practice, at the smaller end of the drum, the method is further defective in two ways: the largest quantity of ore is treated on the smallest screening surface, and the finest sizing is also performed on the smallest part of the drum. If, notwithstanding these inherent defects, the mills can point to the satisfactory quality of their sizing, such results are only to be obtained by the use of long and large sieves, which, compared with trommels of better design, occasion greater cost and increased wear, and require larger space and more motive power.

The various forms of patented sizing apparatus which have been introduced at a number of places avoid several, or even all, of the defects that have been enumerated, but all of them are open to some more or less serious objection which prevents their successful, wide-spread introduction. A number of them, excellent in many other respects, offer far too much

difficulty in repairing. To ths class belong the systems with concentric superimposed or spirally arranged screening surfaces,[1] and all, or nearly all, of the kinds which depend on internal scoops to raise the ore from one compartment of the drum to another. Others have the fault of being too heavy; in their design the ends of each drum are fitted with heavy iron castings for ore delivery, making the whole apparatus very cumbrous, and absorbing too much motive power. Many of the new trommel sets are subject to both of these disadvantages, to which, as a further drawback, an excessive charge for patent royalty often can be added. Probably one of the least objectionable types of trommel classifiers is that known as the Heberle system,[2] for although in this system an inclined axis is generally used, with its attendant disadvantages, the arrangement on the whole is satisfactory. Conical trommels are so set that the ore is charged at the larger ends, and the classifications are made in the order of decreasing size, the coarsest ore-class being the first one separated. By this plan the bulk of the ore diminishes with the size of the ore pieces, so that the finer screens will seldom require larger dimensions than the coarser ones.

Jacketed or concentric double-sieve trommels are hardly ever found in the mills, for the reason that they are inconvenient to repair, except when, through want of space, the use of the simple kind with only one screening surface would be impracticable. In all double trommels the mesh of the inner sieve is made to differ, if possible, by several sizes from that of the outer one, so that it will be much coarser and will certainly not wear out before the mantle needs renewal. The double trommel, then, performs a rough classification, and the ore discharged from it must be subjected to still further sizing.

An improvement in the construction of ordinary trommels, tending greatly to facilitate repairs, has of late been introduced. It consists in setting the sieving surfaces in segmental iron frames, and bolting these with brass nuts from the outside upon the drum spiders. The renewal of the whole surface of a drum can be accomplished in a few minutes, provided that dupiicate segments are on hand.

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[1] Schmidt's Spiral Sieve, Berg u. Huettenm. Zeitung, 1881, vol. xl., p. 108.

[2] Heberle Trommel System: illustrated description, Berg u. Huettenm. Jahrbuch der Bergakademien zu Leoben u. Przibram u. Schemnitz, 1881.

The kind of sizing sieve most widely used is made of punched sheet metal. The perforations are commonly round, though it has been shown repeatedly that sieves with square holes wear as well, cost but a trifle, if anything, more than the ordinary kind, and do decidedly quicker and better work. Wire-mesh sieves are very seldom used, and then only for the finest sizing. Sheet copper is preferred to iron for sieves with holes of 2 mm. ($\frac{1}{12}$ inch) and under, though the practice in this respect is not always the same. Copper cannot rust, so that fine holes in a copper sheet keep open better than in one of iron; but the main consideration in choosing between the two metals is always the relation between cost and wear of the different kinds of sieves, the iron having to withstand abrasion and oxidation, while the copper is deteriorated by abrasion only. At Clausthal all sizing sieves down to the 1 mm. size ($\frac{1}{25}$ in., corresponding to No. 16 mesh) were formerly of iron, but copper has now been introduced for 2 mm. (No. 8 mesh) sieves, for it has been found that the copper sieve, costing there 80 per cent. more than the iron one, lasts over twice as long as the latter. In place of a trommel with iron sieves which need renewal every five months, one with sieves of copper lasting a year has been substituted. For screens of coarser sizes, requiring heavier sheet metal and not so much influenced by rust, there is no economy in employing copper. Steel has been profitably used for 3 and 4 mm. sieves; it wears better and rusts less than iron.

A *regular* feed of ore into the first trommel of a set is deemed essential to good, quick sizing. The most favored automatic devices for securing this end are those which take up the least height. A common form is a rectangular oscillating pan set beneath a pointed hopper. The ore falls upon the pan and is thrown off into the trommel by the shock of the oscillating movement—sixty to seventy strokes per minute. The feed is aided by a jet of water directed into the hopper.

The sizing of seven and a half to eight tons of ore per hour by one set of trommels of ordinary dimensions—the largest trommel being about 3½ feet in diameter and 9 to 12 feet long, according to the number of fields contained—is considered very fair work. When the quantity of broken rock or of finings exceeds this amount, it is treated on two or more sets of sizing drums rather than on one set of larger dimensions. This practice, which is very general, economizes height in compari-

son with what one large set would require, and has the advantage common to most duplications, that in case of a breakdown the whole plant is not so likely to be forced into idleness.

A very common arrangement for a system of trommels is to mount them in steps, side by side, as though set on the flanks of a Λ frame; but a more recent and now favorite disposition places them in line in an inclined plane—one drum in front of the other, and rather far apart, so that the several ore classes formed by sizing can be dropped directly through chutes to their respective picking tables or jigs, and these, when advantageously arranged, have ample free space about them. This disposition has been made in the new dressing works at Lintorf,[1] Rhenish Prussia, where some of the drums are set in the rooftrussing of the mill. Where space is contracted, and particularly where but little height is available, a mode of setting similar to the one already referred to as the Heberle system becomes very suitable. By such an arrangement, in which a number of conical trommels are keyed upon a single inclined axis with the larger end of each drum uppermost, the fall or slope of the sieve bottom will under all circumstances be less than the angle of inclination of the axis. The coning of each drum can be such that the bottom receives a slight dip in the direction of the smaller discharging end, or it may be level, or may even have a gentle rise. The advance of the ore in the trommel depends upon two factors—first, upon the slope of the bottom, an influence which may be naught and sometimes negative, and secondly, upon the zigzag path that is pursued by the material as it rises in the arc of an inclined circle, being held by friction to the side of the trommel, and then falls in a perpendicular plane, on the curve of an ellipse, to occupy a point nearer the discharge end than before and to undergo a repetition of the movements just described. By this system very little fall is required in the trommels themselves, so that most of the pitch can be put into the fixed sheet-iron jackets and troughs, which convey the ore successively from one drum to the next one of the set. It has been found possible to work such an apparatus with a dip of only 17° for the long inclined axis.

By the action of revolving sieves and of rolls and other crushers a portion of every ore, and principally of the soft

[1] Illustrated Description of the Lintorf Works, Engineering, September 30, 1881.

rich mineral it may contain, is reduced to a very fine condition. When the operation of sizing is performed *dry*, this material will adhere as dust to the sands and coarse meals of the ore, and if charged with them upon jigs or into a current of water, much of it would inevitably be lost as *float mineral.* An excellent practice avoids this source of loss at the mills of Przibram, and seems to deserve a more general introduction; it is practically a kind of dry dressing: the fine ore, sized and ready for jigging, is fed continuously on to a long narrow end-percussion sieve, which has a bottom of punched sheet copper with $\frac{1}{2}$ mm. ($\frac{1}{50}$ in.) holes; the sharp shocks given to the apparatus advance the ore upon it and serve to loosen the adhering dust, which works its way down through the ore layer and falls through the sieve. Fine dust thus collected at Przibram assayed 0.014 per cent. of silver (4 oz. per ton), 21 per cent. of lead, and 5.4 per cent. of zinc. When cleansing is combined with sizing, or in any case where the sieving is carried out *wet*, the finely comminuted material is no longer a dust, and therefore not so very apt to be lost as float, but yet if it pass through a succession of operations with coarser ore, and mix with much water before reaching the "slime department" —the place where, according to scheme, it should be collected— there would be danger of losing much of it on the way. Such losses were experienced at Clausthal, where, owing to the very large quantity of ore treated, this item was a considerable one. To remedy the evil all crushed ore of less than 4 mm. ($\frac{3}{16}$ in.) size is now made to pass directly from the coarse sizing drums to fine jigs, which separate *through* sieves of 1 mm. mesh. The fine stuff passing through the sieves is rich, and ready for the smelt works, while the tailings consist of cleaned grit and sands which go on to further sizing.

HAND PICKING.

It has been mentioned that all ore, whether in the form of finings or broken lump, between the sizes of about $2\frac{2}{3}$ and $1\frac{1}{3}$ inches, is usually subjected to hand picking. The principal object in this treatment is to separate finely mineralized pieces of ore from those containing the mineral in more or less compact masses—that is, to sort *crushing ores* from *cobbing ores.* The former, as the name indicates, pass to crushing; the latter to further manual treatment. The operation is generally de-

veloped so as to produce two classes of crushing ores—*roll rock*, with mineral in a moderately fine state of division, and *stamp rock*, in which the mineral occurs in very thin streaks and bands, or in fine dissemination. Each of these classes is further subdivided according to its chemical constituents: in an ore carrying, for example, blende, galena and copper pyrites, pieces with only one of these elements, or those in which one of them largely predominates, are sorted out into separate groups, to be accumulated and dressed separately. In a similar manner several kinds of cobbing ore may be prepared, with a view to simplifying their subsequent hand treatment.

The hand picking of one of the Clausthal ores is selected among many as a fair illustration of the practice; the ore carries argentiferous galena, blende, copper and iron pyrites, and a gangue of quartz and calc-spar. The products of hand picking are:

1. *Cobbing Ores.*—Coarse galena with gangue (to cobbing); coarse blende with gangue (to cobbing); coarse copper and iron pyrites with gangue (to cobbing).
2. *Crushing Ores.*—Moderately rich roll rock (to crushing); poor roll rock (to crushing).
3. *Calc-spar.*—(Sold for gravel.)
4. *Barren Rock.*—(Discarded.)

No stamp rock is here produced, though such a class is found at many other mills. The subdivisions into rich and poor roll rock are intended to counteract the effect of varying quality in the mine output; a supply can always be drawn from an accumulated stock of either kind of ore, so as to maintain a constant grade for the whole product. By this means it is sought to avoid losses which are easily incurred in fine jigging and in the treatment of slimes when there is much variability in the character of the ore.

Hand picking is performed by young boys or girls; the latter have been found the more apt in learning and smarter at their work.

All improvements in carrying out the treatment are made with a view to diminish the amount of handling and to secure better sorting. Most of the new mills are furnished with rotating picking tables. One of these consists of a cast-iron ring, 12 to 15 feet in diameter, constructed in segments and secured by radial arms to a vertical axis. Ore is charged upon the ring at one point and in the course of a rotation it passes

under the eyes of a dozen or sixteen pickers who are seated about the table engaged in sorting. The material which is left by the pickers upon the table—always the largest ore class—is removed automatically by a scraper, while the several sorted classes are thrown forward upon a stationary coned surface which slopes away from the axle of the table and passes beneath the latter to the level of the mill floor. The conical surface is divided off by radial partitions, so as to form a number of chutes, at the lower ends of which are receiving boxes that can be removed when full and changed for others without the least interruption of the sorting. It would be a very easy matter to further develop this system by delivering the several products upon endless bands, which should convey them on to further treatment, so that the handling of the ore be reduced to the one simple act of picking. To secure thorough sorting abundant light is indispensable, and, as a result of experience, the arrangements should be such that the pickers throw all the sorted ore in front of them. The pickers must be held definitely accountable for their work in order to obtain the best results; at some mills each picker has his own receiving box for every class of ore he sorts out, but at other works, where handling is minimized or where very accurate and comparatively difficult sorting is required, the more improved plan is to allot the work of separating out one particular class of ore to one or several individuals, and to hold that set jointly responsible for the quality of their special separation.

COBBING.

The separation of the different minerals of a hand picked ore by means of manual labor and the cobbing hammer is, like the practice of spalling, a legacy of the old-time mill—crude, under many conditions not applicable for the economical and rapid treatment of large quantities of ore, and destined, like its coarser prototype, to occupy a very subordinate position in the list of dressing operations when, as already suggested, mechanical separations are no longer based on specific gravities alone, but the various other physical properties of minerals, their friability and toughness, their change or stability of character under the influence of moderate furnace heat, or their different degrees of magneto-electrical excitation shall be more generally applied in the dressing mill. To-day, however,

cobbing is still an important operation, carried out with a threefold purpose: first, to separate pure masses of different minerals from one another and from adhering pieces of gangue, preparing them at once for the market with a minimum loss from comminution; secondly, to separate several classes of crushing ores according to the predominance of one or the other of the different minerals they contain; and, lastly, to produce a classification according to the nature of the gangue. By carrying out either of the last two separations in the proper cases, the subsequent wet dressing is either simplified or rendered more efficient. The first kind of separation, the direct production of marketable products, seems to be a permanently valuable application of cobbing—that is, the one most likely to preserve its sphere of usefulness. Pure masses of two different minerals which are attached to a piece of gangue can frequently be separated from one another by a single well directed blow of the cobbing hammer, without incurring any loss by the production of dust.

At Hale, in Cornwall, a copper ore, after being sized and hand-picked, furnishes a product for cobbing which carries coarse and fine copper pyrites with some very coarse galena and blende, and fine tin.[1] The classes produced by the hand separation are:

1. Rich copper pyrites (sold to smelters).
2. Coarse rich galena (sold to smelters).
3. Coarse rich blende (sold to smelters).
4. Fine tin (sold to tin dressers).
5. Pyritiferous crushing rock (dressed).
6. Barren rock (discarded).

In this case to preserve intact the lumps of coarse galena and blende, none of the ore is crushed before the hand separation. Galena and cassiterite, with specific gravities of about 7.5 and 6.8 could not be profitably separated from each other by wet dressing if once reduced to a fine condition, and the same is true for the blende and copper pyrites with nearly the same specific gravities. By a hand treatment which is really a combination of spalling and cobbing—since selected lump-ore is treated with both sledge and cobbing hammer—the separations are very easily made. The crushing rock consists almost wholly of copper pyrites and gangue.

The two cases in which cobbing is resorted to for the pro-

[1] *Tin* is a Cornish name for tin ore or oxide of tin.

duction of several kinds of crushing ores—when classifications are required according to predominating minerals, and according to the nature of the gangue—find a good illustration in the practice at Ems, as detailed on page 6.

The extent to which any kind of cobbing is developed depends very greatly, aside from the nature of the ore, upon the cost of labor and the scale by which market prices rise with the increasing purity of the products that are made. As an example of the extreme to which this operation can sometimes be carried with profit, it may serve to instance the Clausthal treatment of one of the cobbing classes produced by hand-picking, and mentioned on page 17 as "coarse copper and iron pyrites with gangue." This ore is separated by cobbing into:

1. Copper pyrites, nearly pure (sold).
2. Copper and iron pyrites—the copper predominating (sold).
3. Iron and copper pyrites—the iron predominating (sold).
4. Iron pyrites, nearly pure (sold).
5. Pyritiferous crushing rock (accumulated and dressed).
6. Pyrites with galena, gangue, and a little blende—separately cobbed, with other material of the same nature, by expert workers to minimize the quantity of dust, and then yielding: galena (smelted); copper and iron pyrites (returned to the first cobbing); pyrites and galena of intergrown texture (accumulated and dressed); galena crushing rock (dressed); pyritiferous crushing rock (dressed); pyrites and blende (subjected to special blende cobbing).

Cobbing is not always one simple and final separation, but at times a succession of operations, as disclosed, for example, in the treatment of the sixth of the foregoing groups, the "pyrites with galena, gangue, and a little blende." At Clausthal twenty-four ore classes are produced during the process of cobbing, but eight of these are intermediate, and only sixteen products leave the cobbing house for the market or for mechanical treatment. The object of subdivisions is to simplify the work for each individual, and to put those separations which require most care or skill into the hands of expert cobbers.

The bulk of ore from the cobbing separation consists always of crushing rock of various kinds. These products are either treated at different times in one mill, or they may go to dis-

tinct and special departments, each furnished with apparatus that is particularly adapted to one kind of material. Many of the large works running on ores of argentiferous galena, blende, and copper pyrites have adopted the latter as an excellent practice. They have one mill for galena and pyrites, and a distinct one for blende—a class of ore which does not, as a rule, require as fine crushing as galena ore, and is often subjected to a very different system of sand and slime treatment, of which more will be said in another connection.

Little or nothing has been done to reduce the cost of handling ore in cobbing. The cobbing tables or benches are usually set along the walls of the mill on the ground level, and the ore is conveyed to and from them in boxes, at a comparatively great expense of manual labor. This arrangement seems unsatisfactory, even where boys' wages are only eight to fourteen cents per day, while in places where the value of labor is higher a profit would undoubtedly be realized in labor-saving devices. It would be easy to arrange a mechanical delivery of the hand-picked ores into hoppers from which the cobbers could draw a supply on to their tables; and, as regards an economical disposition of the products, these could be thrown at once into chutes opening into the cobbing tables, and delivering below on to endless belts, or into bins or cars.

ROLL CRUSHING.

Rolls are used to treat the several classes of *roll rock*, or *crushing rock*, which may result from spalling, hand-picking, and cobbing. The bulk of this ore is in *nut* size, from 1¼ to 2⅜ inches in diameter, though the products from the cobbing table are somewhat smaller.

It is common to find two or three grades of roll crushing in a good mill—coarse, middle, and fine crushing. The ore classes above named are all delivered to the first or coarsest set of rolls. Practice has shown the advantage of crushing, within certain limits, fine and coarse material at the same time, for then the coarser ore, if well mixed with the finer, forms somewhat of a compact or homogeneous mass which is more effectively seized by the rolls than coarse pieces alone would be. It is, therefore, not uncommon to select ore of a comparatively fine size-class—a kind possibly suited for "middle crushing"—and to pass it, together with the coarse roll ore, through the first set of crushers. Fine material, such as is

applicable for this purpose, is found in tailings from the coarse jigs which have been mentioned (while on the subject of sizing—page 9) as treating an ore size that might range down to five-eighths of an inch. The headings or concentrates from those jigs were subjected to fine picking, the tailings to crushing.

It is for experiment to determine to what degree of fineness crushing must be carried in order that the operation may unlock, or free, a sufficient quantity of mineral to render sizing and jigging profitable. Practice in this respect will therefore be as varied as the nature of the ores themselves. In some few cases the crushing is only carried down to 20 mm. (about $\frac{3}{4}$ inch), after which the stuff is sized—for there will naturally be a quantity of material reduced to a condition finer than that required—and then will follow the jigging; but in many cases no hydraulic separation can be undertaken before all the crushing ore has been reduced to at least 8 mm. (about $\frac{1}{3}$ inch). These sizes, 20 mm. and 8 mm., represent approximately the limits within which the reduction preparatory to the first general jigging is usually made, though there is, of course, no reason why the proper size might not be found beyond this range.

Not many ores are found carrying all of their "mineral" in so fine a state of distribution that rolls have to be dropped out of the dressing system altogether. Examples of such a class do, however, occur. By subjecting them to manual treatment several varieties of crushing products are obtained, and these then constitute different kinds of stamp rock. The tin ores of Cornwall and of the Saxon Erzgebirge, and the auriferous sulphuret ores of Schemnitz and other Hungarian localities are some of the principal representatives of this type.

The present tendency to apply rolls for finer crushing than they have hitherto performed manifests itself in a number of mills, where extra roll sets have been introduced, leaving the older stamp batteries to stand partially idle. In no well known dressing works, however, are rolls used for crushing fine ore when *all* of it has to be reduced to less than 4 mm. ($\frac{1}{6}$ inch). Crushing down to 4 mm. will, of course, produce a quantity of stuff of much smaller size, but yet 40 to 50 per cent. of the crushed material will in most cases be of the 4 mm. class, *i. e.*, between 4 and 3 mm. For finer comminution different kinds of grinders have of late years been tried.

Ordinary roll rock is easily reduced to 18 or 20 mm. by a single pair of rolls, but for greater reductions, *e. g.*, for crushing down to 8 or 10 mm., two, or even three successive roll-sets are always used, and it is good practice to interpose between each set, as already observed, a sizing screen which separates out fine sand and meal that would be reduced to dust if allowed to pass to the next set, and which also throws out th ecoarse pieces—those which, owing to the flexible nature of the resistance buffers, have been able to drop through the crushers without a sufficient reduction, and that therefore require to be re-treated by the same set of rolls. The practice of successive reduction is carried out in order to obtain the most satisfactory kind of work from the rolls and to minimize the production of dust. At Przibram, where this system has recently been introduced into a new crushing house to reduce the ore size successively to 22, 14, and 9 mm., one pair of coarse rolls delivers rock to two pairs of middle rolls, and each of these supplies one pair of fine rolls.

The improvements noticeable in modern types of rolls, as compared with older designs, are greater compactness of form and better facilities for making repairs. In some of the latest forms of crushers an exchanging of rolls consumes but very little time; the construction is such that the tensile strains are borne by wrought iron tension rods, and the heavy nuts at the ends of these are screwed up against a cast iron end-piece which bears the immediate pressure of the crushing resistance; in the case of repairs, it is only necessary to loosen the nuts and remove the end-piece, when the rolls can be drawn out of the frame. The handling of the heavy pieces is facilitated by arranging for the ready use of differential pulleys; in one of the mills a couple of these pulleys are suspended from an overhead railway, which is used to run the rolls into an adjoining shop and directly on to the repairing lathes.

Cast iron sleeves and other old devices for coupling the rolls to the power shaft are making way for the simple fast and loose belt pulleys with a speed reduction by gearing.

The superiority of open-hearth steel [1] over cast iron and Bessemer steel as a material for roll tires, has, it is claimed, been demonstrated at several places, and so it has come into use for this purpose. The harder the tire, the better it is

[1] Open-hearth steel is not nearly as expensive, compared with cast iron, as it is in this country; it generally costs about five per cent. more than Bessemer steel.

found to wear but the poorer is the quality of crushing—*i. e.*, the less are the rolls capable of seizing the ore. Steel tires are used until through wear they become very uneven, and then they are turned down for a second run. For example, a tire 3 inches thick and weighing 600 pounds is worked eleven hours per day for about six weeks, crushing hard quartzose rock; then, upon being pared down to one-half of the original thickness and weighing net about two hundred and fifty pounds, it lasts four weeks longer, after which the old tire of 120 pounds is sold as scrap. The practice of removing the tires to turn them down has directed attention to the most expeditious method for securing the tire to the core, or hub, of the roll. The old way, that of driving wooden wedges all around the core, is still popular, but a newer and quicker method—slipping a coned tire upon a conical hub and securing it by bolts—has been introduced at a number of mills.

Rolls for coarse crushers vary from 600 to 800 mm. (23½ to 31½ inches) in diameter, and run at speeds of twelve to twenty-eight revolutions per minute. The best speed for rolls is a matter still under discussion; on the one hand it is claimed that an increase of angular velocity is far from producing under all circumstances an increased capacity, because the rapidly running rolls do not seize the rock well; on the other hand, figures are adduced to prove that great rotary speed is accompanied by an increased output. Experiment readily verifies, within limits, the second assertion, but that a high number of revolutions is therefore an advantage is not a necessary conclusion. At one of the mills where careful trials were made, a pair of rolls, each 650 mm. (25½ inches) in diameter, and with 280 mm. (11 inches) face, had to be run at twenty-six revolutions per minute in order to crush in one hour fifteen tons of hard quartz ore from a two-inch size down to two-thirds of an inch; reducing the speed was equivalent to diminishing the output, but yet, for the most satisfactory crushing—if under this term be included the seizure of the ore as it falls from an automatic feeder, and then its thorough reduction—the maximum speed was sixteen revolutions per minute. High speed and great output in this case involved the extra cost of a boy's labor, for it was found that constant attendance was necessary to shove the ore down between the rolls. This and similar experiments go to prove that for

securing high tangential velocity, which is indispensable to a large capacity, and to provide at the same time that the ore shall be effectively seized by the rolls (even in winter months when ice and snow are possibly mixed with the rock), very much larger dimensions will have to be adopted. Rolls of 1000 or 1200 mm. (39½ to 47 inches) diameter could be employed, and these might, without their action being impaired, be driven at much higher speeds than any now used. The increase of size is only limited by the difficulty and expense in handling and repairing heavy rolls—objections which should be entirely overcome by overhead rails and differential tackle in a new, well designed mill.

Small rolls, 12 to 14 inches in diameter, have been employed for fine crushing in several works, but are now very generally condemned: running very noisily at 60 or 70 revolutions per minute, and with the small ore often dancing upon the faces of the rollers instead of passing through the crushing slit, they are found to have a very small capacity and do not perform satisfactory work. The most approved practice, as now seen in new mills, is to employ rolls of the same size for both coarse and fine crushing; tires while new are used for the fine work, and after their wearing down, which they do with fair regularity (though too unevenly to continue work on fine crushing), they are turned over to the coarse work.

All roll crushers are supplied with an automatic feeder, usually of the shaking tray type. On the discharge end of the tray there is sometimes a triangular tongue, which directs the advancing ore so as to fall upon the sides of the roller faces, and thus save the middle of each face from excessive wear.

Rubber resistance buffers are used almost everywhere; they seem to yield too readily to every hard piece of rock that enters the crusher; the rolls are kept continually moving back and forth, and there is always a considerable class of insufficiently crushed material which requires retreatment. Whether a system of breaking-cups, as applied in some coal crushing rolls, would not serve better than the buffers remains for experiment to determine.

The dust which is unavoidably produced by the action of crushing renders it very desirable to set the rolls, and perhaps the accompanying sizing drums, into a building or section apart from the hydraulic dressing machinery. In all but the very largest mills, however, this disposition has been imprac-

ticable; an economical, and at present a favorite arrangement is to intercalate the rolls in the general sizing system. A long line of trommels, as already described (p. 15), classifies the ore into a number of sizes; the coarsest of these—the picking ore, discharged from the first sieve—is dropped through a chute to the sorting table; it is hand-picked and cobbed, after which the greater portion of it is passed through one or more sets of roll crushers, and finally is raised by a bucket elevator and rejoins the main sizing system. The several classes of jigging ores which result from sizing are subjected to mechanical, or hydraulic, sorting, and usually produce, in addition to concentrates, a number of ore classes requiring further reduction. All of these ore products down to 4 mm. (3/16 inch) are commonly charged to fine crushing rolls which reduce the stuff to 4 mm. and less; this crushed material similarly rejoins the main sizing system at the proper point.

JIGGING.

The operation of sorting ores by jigging is always performed in water; dry jigging has not been attempted anywhere, so far as known to the writer, except in an experimental way. The ores that are subjected to jigging have already been described (p. 9); they are the several size-classes, ranging from 32 mm. down to 1½ or 1 mm. (1¼ to 1/20 or 1/25 inches). A noteworthy exception is found in the very coarse jigging practised in the new Lintorf mill, where two size-classes are jigged between 54 mm. and 30 mm. (2⅛ and 1⅕ inches). This case marks the growing tendency to restrict, when possible, the manual operations in dressing. Only the comparatively small quantity of ore that falls within the limits of 65 and 54 mm. (2⅗ and 2⅛ inch) is subjected at once to hand-picking. The two classes next in size, 54 to 45 mm. and 45 to 30 mm. (2⅛ to 1⅘ inch, and 1⅘ to 1⅕ inch), are treated on three-sieve jigs, yielding pure concentrates and barren rock, and two classes of "middlings," or mixed products, which in the case of the coarser size fall from the jigs directly on to picking tables.

Material finer than 1 mm. is sometimes jigged, though not often; the prevalent opinions on the advisability of such treatment will most suitably be considered under the subject of slime dressing.

The old shaking sieve, operated by hand, developed years

ago into the mechanical jigging sieve, and this has now been supplanted almost everywhere by the piston jig, by sole reason of its better wear. Of the various forms of piston jig—the jig-pump, the jig with jarring sieve, the under-piston jig, etc.,—there remains but one kind in general use, namely, the side-piston jig, a form in which the piston and the sieve are placed in adjoining compartments of the jig-box, with free communication beneath them, so that water can be forced in rising current through the sieve by a descending stroke of the piston. This type answers best to the combined requirements of good sorting, little wear, and simple, easily executed repairs. The other forms of machines may be seen as historical curiosities, or in active use, sometimes, for the careful treatment of small quantities of middlings from the regular jigging work.

Automatic jigs, with a continuous discharge both of concentrates and of barren product, or tailings, are very widely used, though the semi-automatic jigs, from which the concentrates are removed by hand, still find support from a tenacious few, because the quality of their sorting is more perfect than that of continuous machines. This difference in favor of the intermittent jig, however, is very slight, and never able to compensate for the evils of small output and increased mineral loss by attrition, which necessarily attend the semi-automatic type of machine.

The method of continuous working has been attended by a great development of the system of multiple-sieve jigs. Each sieve, however, has its separate piston. The system by which one piston serves a number of sieves has not found favor because a full control of the stroke or water current for each separate sieve did not prove practicable. In a jig with several sieves the light products from the first sieve are delivered by a flow of water on to the second sieve; those from the second are carried to the third, and so on. The number of sieves in one jigging machine depends upon the number and the difficulty of the separations that are to be made. Jigs with two sieves answer well for easy work, such as the separation of hematite (sp. gr. 4.8-5) from a silicious gangue (sp. gr. 2.6)—the object of the operation being to produce at small cost a high grade mineral, regardless of the loss of a few per cent. more or less of iron in the tailings. Three-sieve jigs are now extensively used for coarse jigging when the ore carries two classes of mineral with an ordinary silicious or calcitic gangue.

Four sieves may be employed if the gangue is a heavy one, as spathic iron or baryta, for then the separation requires that the material be subjected to more prolonged jigging action. A jig with four sieves is also adopted when three kinds of mineral have to be separated from one another and from a gangue. For example, an ore carrying argentiferous galena, blende, and iron pyrites in a silicious and dolomitic gangue is treated by coarse jigging on a four-sieve jig which yields:—

1. Concentrates on first sieve, pure galena.
2. Concentrates on second sieve, mixed galena and pyrites.
3. Concentrates on third sieve, pyrites.
4. Concentrates on fourth sieve, blende, with pyrites and gangue.
5. Tailings over fourth sieve, barren rock.

The mixed products (2) and (4) are retreated on the same, or on another, jigging machine, or are subjected to fine picking, or they may be treated by further crushing, according as the nature, quantity, and size of the material seems to warrant. The middlings usually consist of small grains of the heavier mineral, large grains of the lighter one, and medium grains of both minerals with intergrown texture. Sizing might, in some cases, be applied with advantage to this class of material. Commonly, however, it is carefully rejigged on a separate machine, and yields fairly pure concentrates of both minerals, a mixed product, which is again retreated, and one class that goes to crushing.

The middlings, or concentrated product, always obtained from the second sieve in jigging is not unfrequently sold without further purification, but at a less favorable market rate than the purer "heading" mineral from the first sieve.

The concentrated products of the jigs either collect *on* the sieves, to be removed automatically, or they pass through a bed of coarse mineral, and then *through* the sieves, to collect in the hutch-boxes below.

For removing the mineral which accumulates on the sieves there are half a dozen different kinds of automatic discharges in common use. The simplest form, and one which has given very satisfactory results, consists of a pipe of which the upper, open end is flush with the jig-sieve, while the lower end passes through the jig-box, and is partially closed with a slide or a nozzle of given diameter, through which the concentrates can be continuously discharged. The amount by which the slide

is opened, or the diameter given to the nozzle, determines the volume of flow through the discharge pipe, and serves to control the working of the jig. In treating coarse ore sizes—pieces that are more than two-thirds of an inch in diameter—the discharge opening is necessarily a large one. To economize water in this case, and at the same time to guard against too great a flow of ore, the delivery pipe discharges into a stay-box—a deep box of small cross-section, adjoining the jig-box, and preserving with it a common water-level. The concentrates accumulate in this stay-box, and are periodically removed by opening a slide in the bottom. In some mills, however, a stay-box is dispensed with altogether; in treating coarse stuff the lower end of the discharge pipe is kept closed until a comparatively thick bed of concentrates has accumulated, and then the whole quantity is run out at once by opening the discharge vent to the full diameter of the pipe. This practice cannot be recommended; it is impossible to know just when the sieve has been discharged, because the pipe remains, or should remain, full of the concentrate; but it often happens that the discharge valve is not closed until foreign material shows itself mixed in with the discharging mineral. Another objection to this system is that the concentrates are kept longer than is absolutely necessary upon the sieve, whereby the losses from mutual abrasion of the ore particles are increased, just as in the case of the semi-automatic jig.

In using a simple pipe discharge two products of different grades are sometimes taken from a single sieve, the second, or lighter, class being collected by a separate discharge pipe, which passes upward through the sieve and through the layer of the heavier mineral. A two-sieve jig running in this way at one of the mills yields:

1. On first sieve, galena concentrate (sold).

2. On second sieve, *lower* pipe, mixed galena and blende (hand-picked, or sent to special blende dressing, according to size).

3. On second sieve, *upper* pipe, galena and blende attached to gangue (sent to crushing rolls).

4. Over second sieve, barren rock (discarded).

This method is particularly applicable to jigs of one or two sieves when the classification by sorting in existing works is to be further developed, but it is not designed to replace or supplant multiple-sieve jigs in a new mill. If the same material

which was treated in the manner just described were charged upon a three-sieve jig, the new products would be designated in the same way as those above; but product No. (2), the "mixed galena and blende," would probably be more concentrated, *i. e.*, would contain less of product number (3), while the latter, in its turn, would be freer from barren rock. Or, if the quality of the products was satisfactory in the first case, then a greater quantity of material could, in a given time, be separated into those same grades upon the three-sieve jig than would be possible on the two-sieve machine with three pipe discharges.

It is immaterial where, in each division of the jig, an automatic discharge is placed, for concentrates upon the jig sieve will flow in any direction, acting like a liquid of heavy specific gravity. The experience gained in this respect has led to the use of a number of syphon discharges; all of these work well, so far as the delivery of concentrates is concerned, but yet some more or less weighty objection can be urged against every one of the well known forms. They are either needlessly complicated, or require watchful attendance, or create too much dead surface on the sieve—that is, surface not active in sorting—or impair the capacity of a machine by contracting the flow of ore at one part of the sieve.

The plain pipe discharge, as above described, seems to be one of the best forms for the ordinary run of jigging work; it has not, however, been tried with the very coarsest kinds of jigging ore—1¾ to 2¼ inches in diameter—and it is not improbable that, in treating such material, the pipe would easily become clogged, particularly if its form were bent; in this case, therefore, some other kind of discharge would be preferable.[1] One form suitable for coarse ore is known as the Heberle gate; it consists of an aperture in the side of the jig, above the level of the sieve; this aperture is fitted with adjustable slides by which the upper and lower limits of the opening are regulated; a ⊏ shaped piece of sheet iron is fastened as a guard, with its back and sides vertical, in front of the inner face of the aperture, and reaches down as near to the sieve surface as is possible while still leaving room for pieces as large as the particular ore-size treated on the machine to pass between its lower edge and the sieve. The iron guard

[1] In jigging two-inch stuff at Lintorf, Bérard's slide discharge is used, and, as reported, with excellent results.

prevents the light jig products from escaping through the gate, but the concentrates flow beneath it into the small space it encloses, and rising in this space on the syphon principle, a certain quantity of the mineral is discharged at each pulsation of the jig.

The old English practice of discharging concentrates through a bed of mineral and through the jigging sieve was first introduced for continuous working at Clausthal to treat fine material of 2½ mm. ($\frac{1}{10}$ inch) diameter and less; at a later date this method was adopted in a few mills for ore as coarse as 8 mm. (⅓ inch), while more recently at the Lauremburg works, the remarkable practice has been introduced of jigging in this manner all material up to a size of 35 mm. (about 1½ inch).

By utilizing the whole sieve surface of a jig for discharging the concentrate the capacity of the machine can, of course, be very largely increased over that of one which delivers its mineral products over the sieve, but to work the jig satisfactorily by the former system the quantity of the concentrates in the ore, for coarse sizes, should be correspondingly great, for if not baser material can only be prevented from passing through the sieve by operating with a very thick, heavy, and coarse mineral bed, to raise which at each stroke of the machine would require far too much motive power to be profitable. The Lauremburg practice illustrates this point: the coarse ore is not rich, and the quantity of pure concentrates, therefore, not great; in jigging on a three-sieve machine the mineral headings of the first compartment are collected *on* the sieve, but the concentrates from the second and third compartments are a class of roll rock which is large in aggregate bulk, and can be passed *through* the corresponding sieves; the overflow is barren rock, and the whole separation is a good one.

The bed of mineral required for this system of jigging is of a somewhat larger size than the ore class which is treated upon it. The specific gravity of the bed mineral is, as near as practicable, equal to that of the mineral concentrate, though never any lighter; it commonly consists of the same mineral species. The loss of mineral by mutual abrasion of the bed grains is considerable. The beds are stocked anew every two to four weeks, according to the rapidity of wear; the material worn off of the mineral grains is exceedingly fine, and a large portion of it is inevitably lost.

Iron granules, or preferably sharp scrap, as that from the manufacture of punched sheet iron (specific gravity about 7.6) have in some districts successfully replaced galena as a bed material. Iron pyrites, on account of its superior hardness, is used to advantage as a bedding for blende and copper pyrites sieves. Nevertheless, the mineral losses due to the bed and the extra power required to raise the bed at each stroke of the jig are permanent disadvantages of jigging through a sieve, and will probably confine this method of working to fine ore, except in special cases, as that instanced at Lauremburg, where the separation is that of one very large class of concentrates from wholly barren rock.

For jigging ore of fine size the discharge through the sieve is doubtless advantageous. The additional power required to raise the mineral bed is offset by the increase of motive power which the other mode of jigging would absorb in order to force water through an extremely fine sieve, and by the difficulty which is always experienced and the time consumed in sieving through a very fine mesh; the abrasion of the ore bed in the one case is balanced in the other by the rapid wear of very fine sieves, while there remain as advantages for the "through sieve" system for fine work an easy control of the sorting by changing the thickness of the bed, and the fact that jigging through a bed obviates, to a certain extent, the necessity of very close sizing. Of two different sized grains, brought together as equal-falling bodies by a continuous water current, the smaller one will be aided by its smallness in working a way through the mineral bed of a fine jig which the larger one fails to penetrate, and thus a separation may sometimes be easily effected when in jigging *on* a sieve it would have proved very difficult. The superior limit to which jigging through sieve and bed can, in different cases, be carried with advantage has not, to the knowledge of the writer, been as yet determined.

The light ore and gangue product obtained in each division of a multiple-sieve jig passes with a current of water over the dividing dam from one sieve compartment to the next one, and finally the tailings are carried over the last dam and on to an inclined draining screen. When water is to be economized, the tailings are mechanically discharged—particularly those of the coarsest jigs, which would require a considerable volume of water to flow them over the last dam. Rittinger's archime-

dean screw, introduced in Austria for this purpose, is not in general use; a revolving paddle-wheel, or a shovel at the end of an oscillating lever, is the simpler device that is commonly employed, sweeping the tailings over a concavely round dam.

For very fine, or "meal jigging," a large quantity of water is required on the sieves, and also a great number of short piston strokes per minute, in order to keep the fine material from packing. To economize in the use of water, and to prevent the fine material being carried off the jig too quickly, the water in such fine jigs is almost always stayed—that is, the tailings are discharged through a long slit in the end-board of the jig, beyond, and immediately adjoining, which there is a stay-box. The latter may have the form of a small hydraulic classifier, which delivers the heavier material through the bottom, and the lighter stuff as overflow. The overflow level is set at least two inches higher than the discharge slit of the jig, so as to produce a slight head pressure with a tendency to check the main current of the jig. In another form of stay-box the jig discharge is similarly made through a slit a couple of inches below the water surface, while in the box there is a float from which hangs a plug that regulates the discharge opening in the bottom according to the water-level in the jig; the sands and meals sink to the bottom and escape, while most of the water is retained.

Ore jigs are generally built of wood, and are found to last in good condition from eight to ten years, working eleven hours per day. The water used in dressing works is frequently pumped from neighboring mines and is apt to be slightly acid; this precludes the use of iron for jigs, but when such objection does not exist, thin plate iron covered with a heavy coat of paint is being recommended and introduced by large builders of dressing machinery. Experience has shown that iron jigs shake to pieces in a few years if not constructed in the very best manner, particular care being required to strengthen the corners of the jig-box with angle iron.

In continental practice, the piston area is made three-fourths as large, and sometimes fully as large, as that of the sieve, while in England pistons of smaller proportion are often used. The advantage of the large piston is that the jigging is performed with a short stroke and a regular, evenly distributed movement of the water through the sieve. This is held of particular importance in the system of jigging through a sieve and

a bed of mineral to prevent the water forcing its way with great disturbance through certain parts of the bed and the ore charge while other portions of the fine material may be packing. For fine jigs the width of the sieve is not over 450 mm. (18 inches), but for coarse machines this dimension increases to about 550 mm. (22 inches). The usual length of the sieves is from 700 to 900 mm. (28 to 36 inches), dependent upon the difficulty of the sorting; for sieves of 900 mm. the corresponding pistons have two piston rods.

The area of the channel communicating between the piston and sieve compartments of the jig box is never less than that of the piston itself, so that a throttling of the water cannot occur.

A jig box with rounded bottom aids the regular movement of the water, and is sometimes used for fine jigs, even though not applied in coarser ones. In iron jigs the semicircular bottom, which gives to the cross-section of the whole jig box the form of an inverted stilted arch, can easily be introduced.

The piston always has half an inch of play on all its vertical sides; its upper face, when in its highest position, is not more than an inch or two above the level of the sieve, so that it is always covered with water and is never in danger of drawing air.

Jig sieves made of perforated sheet iron and of woven brass wire are both used, with a preference, perhaps, in favor of the latter. The first kind is the cheaper, but offers more resistance to the passage of water, because the flow is contracted by the sharp edges of the holes, which are liable, it is said, to become easily clogged. The sieves are fastened to wooden, or preferably to iron, support gratings of thin, deep design, combining strength with a minimum obstruction of the mesh openings. The sieve mesh for coarse jigs is 3 mm. ($\frac{1}{8}$ inch, or No. 6 mesh), and for finer sand machines 2 mm. ($\frac{3}{32}$ inch, or No. 8 mesh); that for the finest meal jigs is 1 mm. (No. 16 mesh). In this last case a mineral bed of 3 mm. stuff prevents the light jig product, though finer than 1 mm., from passing through the sieve. The principal wear of the sieve, takes place at its head. At one of the large mills, taken as an illustration, a brass wire sieve lasts only eight weeks, when the jigs work ten hours per day on quartzose ore and are hard pushed; the usual run, however, is twelve weeks. At the end of such a period, the head of the sieve is patched, and then it wears from four to six

weeks longer. Each sieve is generally set in a horizontal position, but in some mills, and more especially where water is scarce, the movement of ore along the sieve is aided by a slope of 1:36. In using the Hartz method of jig discharge [1]—the concentrates passing beneath, and the tailings over a dam—it is found necessary to place each successive sieve 2½ inches lower than the preceding one, in order to prevent back currents of water and ore being drawn beneath the apron of the dam which is at the head of each sieve. With jigs having other discharging devices there is no fixed and necessary height between the levels of the several sieves. In coarse jigs each sieve is usually placed from 1 to 2 inches below the preceding one, and the top level of the successive dams between the sieves is lowered in the same degree. In fine meal jigs the difference in levels is less—sometimes ⅜ inch between the first and second sieves, ⅛ inch between the second and third, and nothing between the third and fourth. The object of several sieve levels being only to facilitate the progressive movement of the water and light ore, the drop will, of course, be very much reduced or disappear altogether, as just observed, when very fine material is treated, for this must always be jigged with a very large quantity of water upon the sieves, and the flow has to be stayed rather than furthered in its advance.

The advantage of a short, rapid, rising flow through the sieves, succeeded by a state of quiet water upon them, or, as most, by a gentle descending current, was recognized at an early day. Though experience has since modified ideas regarding the action of the jig, and it is now well known that the rising current which is directly due to the piston-stroke can effect but very little sorting on account of the hindering influence which each grain exerts upon its neighbor, yet a strong, quick, upward current is to-day none the less of importance than formerly. The piston-jig, in its action upon the ore, is very similar to the old movable jigging sieve. A strong rising current, with a sharp, well defined beginning and end, lifts the whole mass of ore almost bodily and unbroken from the sieve; then follows the fall of the material, but, not, as frequently stated,[2] with the velocity of falling bodies in quiet water or in a slowly descending current. Upon the momentary sup-

[1] Illustrated in the Zeitschrift fur Berg Huetten und Salinen Wesen, 1873, vol., xxi.

[2] For example, in Rittinger's Treatise on Ore Dressing, Appendix II., p. 33.

position that the water on the jig-sieve is perfectly quiet, the lowest grains in the mass of suspended ore begin to fall, and, displacing the water beneath them, they create numberless small upward flowing currents which act upon and are reproduced by the succeeding grains, working in this way through the whole thickness of the ore bed, effectually disintegrating it, and affording the bodies a chance to sort themselves in *rising* currents. After this period the ore and the water have exchanged places—the ore, as just explained, has fallen, and the water evidently risen. With the next stroke of the piston and a renewed rise of the ore bed the surface water will overflow the jig dam and carry off some of the ore tailings. Two influences interfere with the efficiency of this sorting, and make it necessary to repeat the operation a number of times: first, the action of each individual grain upon its neighbors, and secondly, the disturbing effect of a downward flow of a portion of the water. Although the ore becomes thoroughly loosened in falling, it can easily happen that a large, heavy grain follows immediately upon a small, light one, hindering, for the time being, the rise of the latter; or that a small, heavy grain becomes wedged between two large, specifically lighter bodies, and is prevented from falling past them. The descending water current caused by the return stroke of the piston can be entirely avoided by the use of a piston with valves in an apparatus such as the jig-pump, but practice has shown that the advantage thereby gained in sorting is, as already stated, more than counterbalanced by the evils of wear and repair which such machines engender. The harmful effect in the ordinary jig can be minimized by giving to the piston a slow return stroke, and by admitting above it an abundant supply of water. The greater the volume of feed water, the less appreciable will be the descending current. But the proper amount of feed water also depends upon the quantity of ore that is treated, so that it is incidentally inferred that a jig should perform the best kind of sorting when being worked to its full capacity—*i. e.*, when there is a continuous stream of tailings passing over the overflow dam; and this, in turn, involves the provision of an ample discharge for concentrates, capable of adjustment for all variations in the richness of the ore.

The earliest mechanical means for securing a slow return stroke of the piston was by the reaction of rubber buffers and

springs; then followed the introduction of floating pistons which were forced downward by revolving cams, and rose slowly by their buoyancy; but it was soon found that all the forms of independent piston for coarse work confined jigging to low speeds and small capacities. Those forms have, therefore, been very generally abandoned, and are now only met with to any extent in England, where the Collom jig is still in use. A more recent slow return movement is that produced by a revolving pinion working in a yoke, which is attached as a variable lever to a rock-shaft that, in turn, actuates a small fixed lever, to which the piston-rod is attached. The principle of the varied motion is that a quick downward movement is performed while the revolving pinion travels over an arc a, while a slow return is made during passage over the much larger arc of $360° - a$, both arcs subtending the same chord on which the length of the piston-stroke depends. A yoke and pinion, however, are subject to rapid deterioration when exposed to so much sharp dust as is usually found in a dressing mill; their place was therefore soon occupied by a more practical elbow-joint mechanism, which is based upon the same principle. One of the forms of this movement now extensively applied is shown in figures 1 and 2. Referring to the illustration, A is a cast-iron frame strapped to the jig-box; B is a uniformly revolving power-shaft, driven by the pulley, C; D is a fly-wheel with a single diametric spoke, to which is bolted a cast-iron sliding piece, E; this piece carries a pinion, F; G is a connecting rod, serving to transmit the circular motion of the pinion, F, to a lever, H, which imparts an angular oscillating motion to the rock-shaft, I. Upon this shaft are keyed two short levers, K, K, which act directly upon the two piston-rods L, L, attached to a single piston 900 mm. long. The figures show the piston-rods in their highest positions and the mechanism set for the shortest possible stroke. When the pinion F is revolved, as the arrow indicates, from right to left through the arc a (136°), the pinion M at the end of the long lever traverses the arc b, and brings the piston-rods to their lowest positions; then, during the whole time taken by F in revolving through $360° - a$ (224°), the lever H slowly returns to its original position. To increase the piston-stroke two adjustments are necessary: the eccentricity of the pinion F must be increased by displacing the sliding piece E, which has for that purpose slotted bolt-holes seen in the figure, and the connecting-rod G

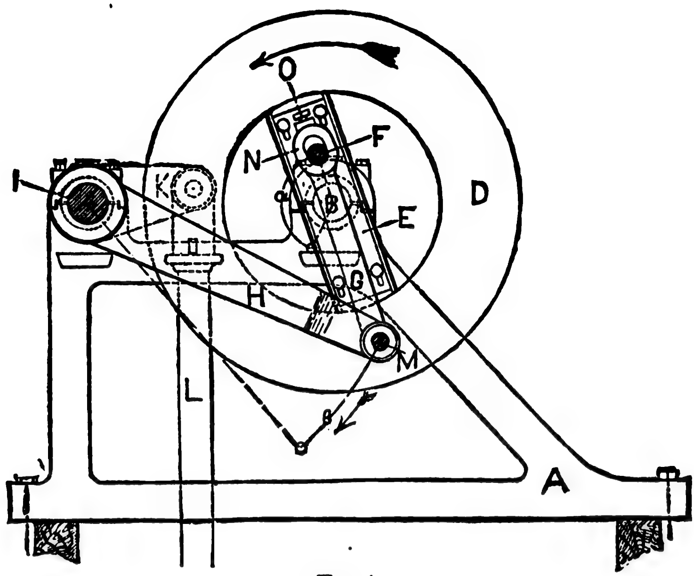

FIG. 1.

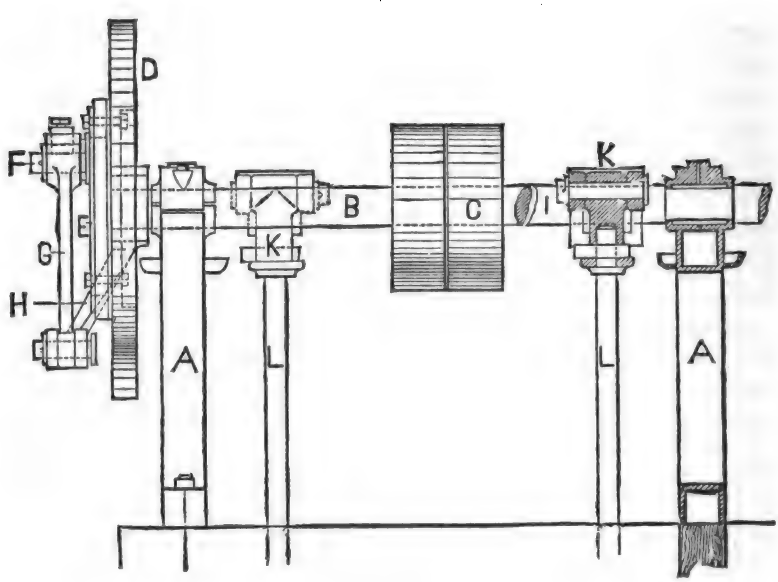

FIG. 2.

must be lengthened so that in its highest position the jig piston shall be at the same level that it was prior to the change of stroke. This second adjustment is made by means of a set-screw O working into the slotted hole which is shown in the boss N at the end of the connecting-rod. The ratio of the duration of time required for the descent and the rise of the piston is in this case as 136 : 224, or as 4¼ : 7; this ratio can be varied by altering the length of the lever H—a dimension which could be made adjustable.

The elbow-joint slow return movement is not favored for fine jigs running at a very high speed and with a piston-stroke of only a few millimetres. It is claimed from experiments, made in quite a number of places, that there is no appreciable difference between the fine separations performed with a slow return and those obtained with the ordinary fine jig movement which consists of an eccentric set on a revolving shaft and directly connected with the piston-rod. Other experiments have not always confirmed this view, however, and the opinion is often expressed that the quality of sorting by fine jigging is still capable of improvement. The stroke of a jig with eccentric movement has been made adjustable by the use of two eccentrics, one being set upon the other, but this device meets with little favor, and even a single eccentric is objected to by some because of the great wear which frictional surfaces suffer from the dust in the mill. Though it may seem that the difficulty of rapid deterioration could be overcome by setting the mechanism inside of the jig-box, immediately above the pistons, and protecting it with a board cover on part of the box, yet a further objection presents itself: the piston movement produced by eccentrics is not the most advantageous one on account of the marked reduction in velocity at the beginning and end of the stroke—the very points approaching which the speed should, if possible, reach a maximum. A good, light and durable mechanism for securing a varied motion in jigging remains to be designed. It is possible that the independent buoyant piston combined with spring reaction, which will permit of high speeds when run on very short strokes, may yet be found leading to the most satisfactory forms.

The length of piston-stroke is 135 mm. (5¼ inches) for the treatment of 2 inch stuff; 90 mm. (3½ inches) for 1½ inch stuff, and 50 to 60 mm. (2 to 2½ inches) for 1 inch stuff. From this ore size downward the stroke decreases about ¼ to ⅛ of an

inch for each successive size until fine *sand* and *meal* sizes are reached, while the strokes of the several pistons of each separate jig sometimes differ within smaller limits. The stroke for ordinary fine jigging work, on ore classes between $1\frac{1}{2}$ and 3 mm., is 9 to 15 mm. ($\frac{3}{8}$ to $\frac{5}{8}$ inch); for meal jigging it is often 7.5 mm. A four-sieve meal jig running at one of the works on blende and galena with a gangue containing spathic iron has a stroke of 7.5 mm. for the first sieve and grades down to 4 mm. ($\frac{3}{10}$ to $\frac{3}{20}$ inch) for the last one. The attempt to separate on this jig spathic iron from blende is not altogether a success. A stroke of less than 4 mm. is never employed.

The speed at which the jigs are run increases as the length of stroke diminishes; it is never less than 75 revolutions of the driving shaft (*i.e.*, 75 jigging movements) per minute for the coarsest work, and it increases for the successively finer jigs by 5 to 10 revolutions per minute, till a speed of 150 or 160 double strokes is reached in the jigging of material finer than 3 mm.: 185 double strokes per minute are sometimes used for meal jigging, and 200 half-inch strokes per minute for the cleansing jigs which separate (as described on page 16), fine mineral from *pea* and *nut* ore. Higher speeds—reaching 300 revolutions of the driving shaft per minute—are advocated for fine jigging by specialists in ore dressing, because the jig capacity is thereby increased. To preserve good sorting this fact must be duly considered in determining the various proportions and adjustments of the machine; neglect in this respect will account, it is claimed, for several failures that have been announced as the results of speeds exceeding 200 double strokes per minute. Whether there is really an advantage in such speeds for any kind of fine ore is not yet a matter beyond dispute and investigation.

It is well understood that the sorting action of a jig depends upon the length of the piston-stroke, the number of strokes per minute, the supplies of ore and of water, the discharges of concentrates and of tailings, and, when working through sieves, upon the thickness of the mineral bed.

After a suitable stroke and speed have once been selected they are usually maintained without variation. Only a radical change in the character of the ore calls for an alteration in respect to these points. By way of example of such a change, a new mill at Grund in the Hartz region dresses a galena ore with a gangue of slate and heavy spar (baryta, specific gravity

nearly 4.5) that are not much intergrown. The ore is separated by hand treatment, as far as practicable, into two classes according to the two different gangue minerals. In jigging the slate ore a little heavy spar is obtained with the galena concentrates and is removed by repicking. The heavy spar ore is accumulated and dressed periodically; on these occasions the strokes of all the jigs are increased to their maximum extent, or by about thirty per cent., while the speeds are reduced by ten to twenty per cent., and a very good separation is effected.

The discharge of tailings, so far as this is connected with the design of the jigs, is generally constant, or but indirectly influenced by the action of a stay-box, as already explained (p. 33).

There remain then, ordinarily, as variable factors for easily controlling the operation, the ore and water supplies, the mineral discharge, and, in fine jigging, the thickness of the mineral bed.

The ore supply is increased with the ease of separation, so that there is no one fixed capacity for any jig. The automatic discharge of concentrates from the sieves is increased with an improving richness of the ore, and the water supply increases both with the quantity and the improving quality of the ore treated—with the latter, because in such a case a greater volume of water will escape through the necessarily enlarged mineral discharge aperture. In jigs which work the concentrates through the sieves, these products settle in each jig-box, and, by the opening of a nozzle, are periodically washed out with a great flow of water which, for the moment, drains the sieves. There is in this case, therefore, no possibility of regulating, as in the other class of jigs, the degree of concentration of the mineral by a continuous flow of water through a mineral discharge aperture, but the same control is attained by variations in the thickness of the mineral bed upon the sieve. No figures for an average thickness of the bed can be given; it ranges between 20 and 80 mm. (⅘ and 3⅛ inches) increasing with the difficulty of the separation and the poorness of the material treated. The ore is carried by the water current over the mineral bed in a layer 30 to 100 mm. (1¼ to 4 inches) thick. The tendency of the high speed advocates is to the greater thicknesses both of ore and bed.

A two-sieve jig with sieves 450 mm. (17¾ inches) wide sorting a ¾-inch (1 inch mesh) size-class of easy dressing ma-

terial, has been found by experiment capable of treating half a ton of ore in five minutes when worked to fullest capacity. This performance would bring the capacity of the machine up to 60 tons per day of ten hours; but such a result is never realized in practice. The ordinary capacity varies from 20 to 36 tons, with an average requirement of 2 cubic feet of water per minute for each sieve. The principal cause of the low efficiency lies in an irregular or insufficient ore supply. The coarser the jig, the greater as a rule is its capacity. The thickness of the ore layer upon coarse *pea* and *nut* sieves varies with the difficulty of the sorting, and accordingly is found ranging between the limits of 120 and 200 mm. (4¾ and 8 inches).

Fine "through sieve" jigs working on *sand* sizes treat 12 to 18 tons per day with an average water supply of 1 to 1½ cubic feet per minute for each sieve, while the daily capacity of *meal* jigs is, under favorable conditions, at most 8, or rarely 10 tons (dry weight).

An easily controlled automatic ore feed is used at some works, even where the ore falls to the machine directly from the sizing drums. A feed hopper will accumulate a small supply of ore and counteract the irregularities of working; notwithstanding this precaution, however, the supply often becomes exhausted and the jigs now and again run idle for a short while.

Chips of wood are always present in quite considerable quantities in the ore. A plain strip of metal cut from an old sieve and placed across each jig near the overflow, serves as a skimmer to collect these chips and prevent their passing on with the tailings to fine comminution, and eventually to clogging in the finest screens.

The arrangement of jigs with reference to the economical handling of the products is, with a few creditable exceptions, extremely poor. The machines are generally placed on the ground floor of the mill, and both concentrates and tailings accumulate in collecting boxes, out of which they have to be raised by shovelling, involving considerable expenditure of manual labor. In a few mills, however, the jigs are set six or eight feet above the floor level; and the products fall over inclined draining screens into bins, from which they are drawn into cars, while only the drainings pass to a system of settling tanks in the floor of the building. The cheap price paid

for the labor of young boys who attend to jigging has in many cases caused good mechanical arrangements to be overlooked.

Many ores which are crushed, hand-sorted and jigged do not carry over twelve per cent. of galena and 2½ ounces of silver to the ton, and in such material the portion which reaches fine *sand* and *meal* jigging is often but one-third as rich as the undressed ore.

Rough Hydraulic Separation.

The mechanical sorting of screen-sized ore classes by means of jigging is almost universal, and yet in a few cases in which the ore consists of a single mineral species that is very easily separated from its rock gangue, no such accurate machine as the jig is required. A very interesting ore of this kind occurs at Bleiberg, near Commern, in the Eifel Mountains, west of the Lower Rhine. It consists of a very friable, white, Triassic sandstone, through which are disseminated small nodular particles of galena cemented to quartz. Each nodule carries twenty to thirty per cent. of lead, while the ore, as a whole, runs two per cent. of lead. The mechanical treatment consists, first, in separating the nodules from the sandstone, and next, in stamping the freed nodules and washing the resulting slimes to raise the proportion of lead to sixty or sixty-five per cent. The first separation is a very easy one; the soft ore is often hardly more than a compact sand; ninety per cent. of it is sized without any preparatory crushing; the larger sizes are hand-sorted and jigged, but the bulk of the ore, which is smaller than 6 mm. (¼ inch), is treated on an automatic syphon separator which is remarkable for its simplicity and great capacity. The products from this apparatus are: *Overflow*, sand which is discarded; *Precipitate*, nodules of cemented galena and quartz, and some fine sand; this product is sized on a 1¼ mm. sieve, which yields *over* the sieve, material for stamping followed by hydraulic classification and slime washing, and *through* the sieve, material to be treated on a second syphon separator; this is turn produces a light overflow which goes to stationary round tables, and a precipitate that is treated on four-sieve jigs.

The syphon separator[1] in its simplest form consists of a

[1] The Syphon Separator: Henry, *Préparation Mécanique*, Ann. des Mines, 1871, 6 sér., tome xix., p. 355, with illustrations.

Illustrated description of the Separator, and review of the Mechernich processes

deep, narrow box with two compartments which communicate to form an inverted syphon. One compartment receives a continuous supply of water; upon the water surface there is a float, connected by a rod to the movable end of a single-armed lever which is set above it. The second compartment is furnished with a funnel-shaped false bottom, consisting of a sieve with finer mesh than the ore treated; in the apex of the funnel is an outlet pipe, controlled by a plug stopper which is suspended from the same lever that is connected with the float. The ore is fed continuously into the second compartment of the apparatus, and settles on the funnel sieve. Water coming from the first compartment passes in ascending current through the false bottom, raising the gangue matter and discharging it over a dam. The mineral accumulates on the sieve, soon impeding the passage of the rising current. It acts like a liquid of heavy specific gravity in one arm of the syphon; the water level therefore rises in the other arm, carrying the float with it; by this movement the lever is raised and draws the suspended stopper from the neck of the funnel, allowing the mineral to escape through the outlet pipe. The float then falls and the operation is repeated. This simple apparatus treats sixteen hundred tons of sand in twenty-four hours, with a consumption of one-fourth of a cubic metre of water per minute, or sixty gallons per ton of ore. Its introduction makes it practicable to work four thousand tons of ore per day at small expense, and so realize profits on very low grade material. The action of the separator is the same as that of a spitzlutte, with advantages over the latter, for this particular work, of a smaller consumption of water and an automatic regulation according to the varying size or quality of the ore.

REVIEW OF COARSE DRESSING AND INTRODUCTION TO SLIME TREATMENT.

The mechanical treatment of ores from *lump* size down to one and one-half or two millimetre *sand* grains presents on the whole, in its newer phases, only that variety in detail which is due to differing qualities of the ores, and to various local and pecuniary considerations. Wherever there is considerable difference between the specific gravities of the minerals to be separated (*e.g.* galena 7.6, and pyrites 4.8 to 5.1) future advances

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of ore dressing and smelting: Engineering and Mining Journal, 1877, vol. xxiii., pp. 121 and 136.

in this field lie not so much in the direction of further concentration, or of reducing mineral losses, as in a greater development of continuity between the processes and in mechanical improvements of the individual types of apparatus. In cases where the specific gravities of the various minerals differ but slightly (*e.g.* blende, 3.9 to 4.2; iron pyrites, 4.8 to 5.1; copper pyrites, 4.2; spathic iron, 3.7 to 3.9, and baryta, 4.3 to 4.7), the ordinary methods of separation involve far too much manual treatment; recently, however, better mechanical processes have been devised for such separations, and judging from the work actually accomplished, they give promise of securing more general recognition.

The treatment of ore in fine sizes and finely mineralized offers no such encouraging prospects: opposing opinions on the subject of slime washing and correspondingly diverging modes of treatment are found in many of the mills, and it is but too evident that a broad theoretical basis upon which to develop sound practice is wanting in this department. With the single exception of new, successful methods of comminution, no radical and completely satisfactory advance in the system of slime dressing has been made in foreign mills for a long time; not that such advance has ceased to be a great desideratum, for the losses in treating slimes continue to be high, but the problem is one which still seeks a solution. The only improvements as yet to be noted in slime concentration are more or less successful developments of old methods, so as to apply in newer apparatus the economies of continous working, and to effect large, though still far from satisfactory savings of fine mineral.

The principal operations in slime dressing are the comminution of finely mineralized ore, the hydraulic classification, or grading, of the resulting slimes, and the hydraulic sizing of the graded material by means of buddles, tables, etc. This order, developed in practice, will be adhered to in discussing the operations, though occasional digressions will be necessary for the purpose of explaining the rationale of some particular treatment by an immediate consideration of the operations which succeed it.[1]

[1] The word *slime* is here used throughout in a generic sense, as a rendering for the word *Truebe* of German ore dressing terminology; it comprises all classes of material commonly treated in the "slime department" of a mill—from *fine sand*, 1 mm. or 1.5 mm. or even 2 mm. in size, down to the finest *pulp*.

COMMINUTION.

The ore classes with finely divided mineral, from hand sorting and jigging, are subjected to comminution to unlock, or release, their constituents preparatory to further concentration. Drop-stamps are still the most common kind of apparatus used for this work. In order to supply the stamps with the kind of material that is most easily worked, the fine stamp rock from jigging is always mixed with coarser stamp rock derived from spalling, hand-picking and cobbing. The size to which the ore is to be reduced by the comminution depends upon the fineness of its mineralization. Stamping is usually carried to 1 or 1½ mm. ($\frac{1}{25}$ inch), but a large portion of the rock is always unavoidably reduced to a finer condition. There is an interesting practice at Schemnitz, Hungary, in treating a quartzose ore of argentiferous galena and copper pyrites with finely disseminated free gold. The ore is crushed under light wooden stamps to 2 mm. ($\frac{3}{32}$ inch), and then classified in spitzlutten—or syphon V-troughs—and concentrated on percussion tables; the "heads" are galena and pyrites; the "middlings," which are coarser but specifically lighter than the heads, consist of quartz carrying free gold and fine galena and pyrites, while the "tailings" are very fine galena and pyrites with quartz pulp. The middlings are re-stamped to 0.4 mm. (about $\frac{1}{60}$ inch), and passed through amalgamation pans to obtain the gold, after which the stuff is washed on various slime tables. By this method much less galena is lost than would be the case if all the ore were crushed to 0.4 mm. in the first reduction.

The principle of building stamps with high drop and high speed, involving the use of single or double-toed cams and a large discharging capacity, is being gradually adopted, though there are many mining regions in which the old-fashioned, low-speed stamp battery is still in use.

The ore delivery into the battery is always automatic, so that one man is able, in a large mill, to take the charge of eighty to one hundred and fifty stamps. The self-feeder is actuated either by a blow upon an elbow-lever from a special tappet on one of the stamp shafts, or, preferably, by an independent cam or eccentric. A quick and very short reciprocating motion for the feeding trough has not been found as satisfactory as a slow motion; with the former the fine ore is apt to pack in large supply bins. A portion and sometimes

all, of the stamp water is used to promote the discharge of the feeder. The ore is usually fed to the middle stamp of a battery, the order of stamp drops being such that the middle stamps tend to drive the stuff towards the ends of the mortar, while the end stamps send it back again. A supply of ore given to each separate stamp would probably increase the capacity of the battery and could readily be produced by automatic means. The cam-shafts of the batteries are not, as a rule, set very close to the stamps; a deficiency, therefore, in the ore supply and the consequent striking of a stamp shoe upon the mortar die, does not, as in the California stamp, involve the danger of injuring any of the cams by the low level of the falling tappets, and hence a very careful ore delivery does not seem to be considered of vital importance.

An accumulation of ore under the end stamps of an ordinary battery has been prevented at the Clausthal dressing works by removing, as far as practicable, the objectionable ends—that is, by increasing the number of stamps in one battery from five to eleven heads, which receive the following order of drop: 6, 2, 10, 1, 11, 3, 9, 8, 4, 7, 5. The Clausthal stamp shoes are of tough steel, rectangular, 120 x 180 mm. (4⅝ x 7⅛ inches), and 380 mm. (15⅛ inches) high; they are set very close together, so that the stamp stems are only 125 mm. (5 inches) from centre to centre, and the battery, notwithstanding its large number of stamps, retains a compact form. The cast-iron die is in one piece, and has a face half an inch broader than the length of the shoes; this proportion has been found to secure the most regular wear both of the shoe and die, without incurring the formation of deep pockets in the die and the attendant production of fine pulp. A shoe lasts a year (working twenty-two hours per day); while the die is worn out in ten to twelve weeks. The weight of one stamp is 200 kg. (440 pounds) and the drop is 200 mm. (8 inches); the speed at which the stamps are run is 60 drops per minute, and the maximum capacity of an eleven-stamp battery including stoppages for repairs, is 15 tons of hand-sorted and of jigged quartzose stamp-rock, which is crushed down to 1 mm. in twenty-two working hours, making an output, therefore, of 1⅓ ton per day for each head.[1] An inspection of old shoes and dies showed that their wear was in many cases just as regular as that often observed in the correspond-

[1] Under ordinary conditions, without observing special care in feeding, the running capacity of a battery is not over 10–11 tons per day, or barely 1 ton per head.

ing parts of the rotating California stamp. The compactness of the battery, and the small amount of lost space in the mortar, are strong recommendations for this form of the gravity stamp mill with close set, non-rotating stamps.

The method of simple discharge through a sieve has been tried and discarded on account of the large proportion of needlessly fine slime which is thereby produced. The "stay-battery" is the generally preferred form; it effects the discharge from the mortar through a sieve into a stay-box, out of which the slime issues through a plug-hole with twenty to twenty-four inches of hydrostatic head. The battery discharges upon one side only, and the sieves have retained the old vertical position, which, in a stay battery, seems fully as good as an inclined setting. The stamp-shoes are not lifted above the water level in the mortar, so that splashing is avoided, while a free circulation of water exists on both sides of the sieve, promoting the discharge of slime from the mortar and keeping the sieve surface clean.

In order to reduce the amount of "dead stamping"—or needlessly fine comminution—the lower edge of the sieve is set on a level with the upper surface of the die, or dies, in the mortar. To preserve this common level, several sieves are sometimes used; each one is set into a separate frame, and the lower sides of the frames are all of different width. By the successive use of the proper framed sieves, a means is afforded of lowering the sieving surfaces as the dies wear down.

Woven sieves of brass wire are commonly preferred to punched sheet iron sieves in the mortars, because they offer a greater discharging area.[1] They need slight repairs after the first fortnight of wear, and last, with frequent patching, from two to three months.

The water consumed in wet stamping varies with the degree of reduction; the coarser the slimes are to be, the quicker must the ore escape from the mortar, and hence the greater will be the flow of water through the battery. At Clausthal, an eleven-head battery, crushing to 1 mm., requires 7½ cubic feet of water per minute, or 9,900 cubic feet per day of twenty-two hours, making 660 cubic feet (equals 20.4 tons of two thousand pounds) or 4,400 gallons per ton of ore. At different mills, the water supply for a stay-battery is found to vary from

[1] In Cornwall, punched copper sieves with thirty-two to thirty-six 1-mm. holes per square inch are the general use.

3,500 to 9,000 gallons per ton of ore which is stamped to 1 mm.; an average is about 0.6 cubic foot per minute for each stamp-head, which makes 4,600 gallons, or 21⅓ tons of water per ton of ore.[1] These figures argue in favor of the stay-battery, for the quantities of water consumed are not large considering the coarseness of the stamping; when the reduction is carried to the size of fine pulp, experiment and a limited practice have shown that the supply of feed water can be reduced by fully one-half, without impairing the crushing capacity of the battery for that particular kind of work.

In a few stamp batteries of recent construction the frame, mortar, etc., have been built of iron, and the shoes and dies are of steel, which is very tough rather than hard. The efficiency of iron as a structural material for batteries is still regarded as a mooted question.[2] At Przibram a stamp battery has been erected which is characterized by the introduction of slotted stamp stems; the cam-shaft is placed very close to the stems, and the cams, in revolving, pass through the slots. The purpose of the design is to bring the lift as near as possible into the line of gravity of the stem, and so reduce the requirement for motive power. Another battery has been erected which has rotating stems—a slightly modified design of the California stamp. Very careful experiments were made with these two forms of battery,[3] the result being that for a given amount and kind of crushing the stamps with slotted stems require barely over three-fifths as much motive power as the rotating stamps. The final judgment in the trial, however, was in favor of the rotating stamps, because their capacity was twelve to twenty-five per cent. greater, and they caused less wear of the steel shoes and dies.

[1] In good American practice, the water supply averages 0.5 cubic foot per minute for each stamp in crushing to a fine pulp ($\frac{1}{60}$ to $\frac{1}{100}$ inch), which would amount to nearly 3,000 gallons, or 13¾ tons of water per ton of ore, allowing a crushing capacity of 36 hundredweight per head in twenty-four hours.

[2] In home practice, five-stamp battery frames, built of channel iron, firmly and carefully braced, have rendered excellent service, particularly in districts difficult of access, or in prospecting work, but nothing heavier than 250-pound, or, at most, 300-pound stamps are used. The attempt to employ heavier stamps has shown in a number of cases that the bolts and fastenings of iron frames are loosened by the great degree of vibration and that the iron construction rapidly deteriorates.

[3] Description of the stamps, and details of experiments: Berg u. Huettenm. Jahrbuch der Bergakademien zu Leoben u. Przibram u. Schemnitz, vol. xxix., 1881.

Illustrated description of stamp battery with slotted stems: Oest. Zeitschr. fur Berg u. Huettenwesen, vol. xxviii., p. 336, 1880.

Pneumatic stamps have been used with success in England. In one of the prevailing forms a horizontal crank-shaft moves a number of air cylinders up and down; in each cylinder there is a solid piston to which a stamp is attached by a long stem. When one of the air cylinders is in its lowest position the corresponding stamp rests upon the mortar die, and its piston occupies the middle of the cylinder. As the cylinder rises, the air below the piston is compressed and the stamp is jerked up; during the reverse stroke of the cylinder, compression takes place above the piston with the effect of driving the stamp down with greater speed and force than a simple drop-stamp of the same weight would attain. There are two rings of apertures near the middle of the cylinder to allow the escape of air which is displaced by the actual motion of the stamp, so that the blow is never weakened by cushioning.[1] Each stamp strikes 150 blows of 15 inches drop per minute; the revolving cranks have 5 inches pitch. A pair of stamps crushes 10 to 13 tons of hard tin ore in ten hours, or fully ten times as much per head as the old Cornish drop stamps, with a diminished expenditure of fuel per ton. The practical results at several mills running on very hard rock have shown a consumption of 40 to 60 pounds of soft coal per ton of ore. The increased output has, to a certain extent, been found proportional to an increased striking surface of the stamp shoe. The first cost of a plant with pneumatic stamps is said to be 60 per cent. less than that of one with the ordinary Cornish drop-stamps of the same crushing capacity. The principal influence operating against the more general introduction of the pneumatic stamp has been the cost and frequency of repairs. Several causes of wear have been eliminated in the English stamp by playing cooling jets of water upon the stamp-rods, and by placing the air cylinders so high above the mortar as to be beyond the reach of splashing grit and slime. As soon as the durability of the pneumatic stamp in its present, or even in a still more improved form, becomes satisfactorily established, this kind of battery will merit an introduction wherever stamps are to be used for crushing, except, perhaps, in mining regions which are very difficult of access.

When ore is to be crushed to the size of *fine sand*, and not

[1] Illustrated description of the pneumatic stamp, with details of working: Transactions of the North of England Institute of Mining and Mechanical Engineers, vol. xxx., p. 139, 1880–81.

to the condition of *meal* or *pulp*, even the best stamp batteries are very objectionable, because they reduce by far the greatest proportion of the ore—frequently over ninety per cent.—to a much finer condition than is required. Pulp slime is very unsatisfactory material to treat on account of the imperfect methods which prevail, and its presence in large quantities adds greatly to the cost of dressing. Owing, moreover, to the generally soft or brittle nature of the "mineral" as compared with the ore gangue, the pulp is frequently richer than the sand-slime; in some cases it runs four times as high in mineral as the latter—a condition which tends to increase all the more the losses in washing. The successful design of new forms of comminuting apparatus, which perform satisfactory crushing without the production of much fine meal and pulp slime, is therefore an advance of permanent value in ore dressing, and that it is regarded as such the rapid introduction of the new machines and the discarding of stamps in many of the works attests.[1] None of the earlier crushing mills—the Carr disintegrator, the Motte, the Vapart, and the Ball mills, and many others in which the action depends upon the friability of the ore—have come into common use for ore dressing purposes, except for certain special separations, of which examples will be given elsewhere. Many of these apparatus prove more or less satisfactory for reducing or pulverizing coal, phosphates, flint, cement, etc., but they cannot crush ore to sand and coarse meal sizes without producing much pulp; the degree to which they comminute admits of no sufficiently definite regulation, and, if applied to hard ores, most of them absorb much power and suffer great wear. The newer machines are more positive in their action, and less subject to rapid deterioration. Among the leading types are the Dingey pulverizer, the Heberly mill, the Brink and Huebner disintegrator, and the Schranz roller mill.

The Dingey mill is an English machine which has repeatedly been described.[2] It consists essentially of a horizontal, slowly rotating, shallow pan, the vertical rim of which contains

[1] An interesting comparison between stamping, fine rolling, and grinding is found in an article by Habermann: Jahrbuch der Bergakademien zu Leoben u. Przibram u. Schemnitz, vol. xxx., 1881.

[2] The Dingey Mill; illustrated description, with a series of comparative tests for crushing effect and motive power; Oest. Zeitschrift f. Berg-u. Huettenwesen, Nos. 23 and 24, 1878. Cornish Experiments, illustrated; Trans. North of England Inst. of Mining and Mech. Engineers, vol. xxx., p. 142, 1880–81.

sieves. Four horizontal, annular disc runners are set close down upon the bottom of the pan, and revolve at a rate of two hundred revolutions per minute. Ore is fed with water through the open centre of each disc, and is then caught and ground between the flat surfaces of the discs and the bottom of the pan. Radial furrows in the pan bottom and spiral grooves in the runners aid the passage of the ore through the grinding space and convert a part of the comminuting action into shearing. The crushed material, acted upon by centrifugal force, flies against the sieves, and, if sufficiently fine, passes through them. The direct wearing surfaces of the runners and the bottom of the pan are fitted with replaceable shoes of chilled Bessemer steel or of chilled cast iron ; the latter has been found more durable than the steel. Though the sieves prevent over-sized particles passing out of the mill, experience has shown that they impair seriously the capacity of the apparatus, and hence they are not employed in the Neuerburg mill, which is another machine of horizontal form.

All machines hitherto designed with horizontal runners seem to have certain common disadvantages: the slightest want of symmetry or equilibrium in the runners is productive of irregularities in the wear which it is impossible to overcome, and the degree of comminution is not under perfect control. The crushing pressure is produced by the weight of the runners upon the ore, and although it might be increased by the use of springs, the pressure can never be practically diminished (for the treatment of very friable ores or for coarse work) to less than that fixed weight.

Machines of later design have been constructed on the same principles as the Dingey pulverizer, but with vertical runners and one or more vertical revolving plates. The best-known and most commonly used machine of this kind is the Heberle mill.[1] It consists of a large plate slowly rotating on a horizontal axis, and of two small rapidly revolving runners, each having a plane annular grinding face and a coned centre. Both of the runners are set on the same side of the turning plate, very close to and parallel with it, and in the fields of its

[1] Illustrated description of the Heberle Mill: Berg u. Huettenm. Zeit'g, vol. xi., p. 400, 1881.

Illustrated description and comparative experiments between the Heberle and Dingey Mills: Oest. Zeitschrift f. Berg u. Huettenw., vol. xxviii., pp. 384 and 581, 1880.

lower quadrants. The large turning plate is pierced by a number of holes set radially and in a ring, so as to form in it a circle of apertures. Ore supplied by launders to one side of the plate passes, during the revolutions of the latter, through the apertures, and falls directly upon the narrow slit between the inner edge of the grinding face of each runner and the wearing surface of the plate. Seized by the runners and partly ground and partly sheared, the ore is reduced, and drops out through the bottom of the machine. The closeness of the runners to the plate and the pressure they exert upon the ore can be accurately adjusted by hand-screws and rubber resistance buffers. The wearing parts of the machine are made so as to be easily replaceable; shoes on the runners last for 80 twelve hour shifts.[1]

Material varying in size from 3 to 10 mm. ($\frac{1}{8}$ to $\frac{3}{8}$ inch) is reduced by grinding machines to a fineness of 1 to 2 mm. ($\frac{1}{25}$ to $\frac{1}{12}$ inch) and the reduction has sometimes been carried down to 0.75 mm. The machines work to poor advantage and with very small capacity on ore above 4 to 5 mm. ($\frac{1}{6}$ inch) in size. As the result of long continued working on quartzose ore at Przibram, the quantity of material crushed down to 2 mm. ($\frac{1}{12}$ inch) by a Heberle mill with two runners was;

2,466	pounds of	4	mm. ore	per hour
1,368	"	6	"	"
1,157	"	9	"	"

this capacity proved to be double that of a Dingey mill of similar size running on just the same material. In reducing 2 mm. stuff to 0.75 mm. the capacity of a small Heberle machine with a single runner was 600 pounds per hour. The comminuted material from all the crushing mills which discharge without intervening sieves is usually passed over a screen to separate out the particles which are not sufficiently reduced. Experiments with the Heberle grinder fix the proportionate amount of this class of material at about one-third of the whole quantity of ore treated. This proportion could easily be reduced to a more favorable figure, but by so doing the percentage of fine meal and pulp would be much increased. The working results, above given, represent the net capacity of the

[1] The Bogardus mill is a grinding machine of the same type as the Heberle mill, over which several advantages are claimed in details of construction and design of the wearing parts.

machine. The proportion of sands and coarse meals produced by the comminution tabulated above, amounted respectively to 70%, 62%, and 58% of the completely reduced ore, the remainder in each case being fine meal and only a small percentage of pulp. In other tests the ore, after being crushed to 2 mm., consisted of 91% to 92% of coarse slimes, the balance being fine meal and pulp. Much of the "mineral" is found, after comminution, in the coarser material, in particles of 0.5 to 0.75 mm. diameter, with sharp edges and corners, showing that they have not been subjected to hurtful triturating action.

A Heberle machine at Przibram has been found to consume, for the most favorable conditions of working, 5.4 gallons of feed water per minute for each runner, making 528 gallons of water per ton (2,000 pds.) of 4 mm. material, which is thoroughly reduced, and 1,120 gallons per ton of 9 mm. ore. The small consumption of water for wet crushing in grinding machines is an important consideration in their favor, not only on account of the direct economy in water, but because it enables dressing works to dispense with elaborate slime concentration boxes.

The Brink and Huebner disintegrator,[1] a machine with rapidly revolving discs armed with studs, performs crushing partly by impact and in part by grinding, and is claimed to treat double as much ore as the Heberle mill, and three to four times as much as a Dingey pulverizer. A first experiment verifies the large capacity, but this diminishes considerably with the rapid wear of the studs, showing the importance of not limiting a practical test of this kind to a short trial.

Comparative experiments made at Przibram between different comminuting machines established the following results, which represent the average of a year's work:

Comparison for	Order of Efficiency. 1st, 2d, 3d, or 4th, for the			
	Heberle Mill.	Dingey Mill.	B. & H. Disintegrator.	Fine Rolls.
Minimum production of fine pulp....................	1	4	2	3
Labor, power, and lubrication.	4	3	2	1
Wear per ton of ore.........	1	3	4	2

[1] Description of the Brink and Huebner disintegrator, and comparative experiments with other crushing machines: Jahrbuch der Bergakademien zu Leoben u. Przibram u. Schemnitz, vol. xxx., 1882.

For a given amount and kind of crushing the Brink and Huebner disintegrator produces thirty per cent. more fine meal and pulp slime than the Heberle mill. The item of wear given in the table proved to be respectively 9, 22.5, 36 to 48, and 8 to 14 cents per ton of ore, figures which testify against the Dingey mill and the disintegrator. The durability of the latter could be improved by the use of studs of very tough steel, and then this machine, with its large capacity, would probably be well suited for treating ores of medium hardness, such as those with calcitic gangue.

The Schranz roller mill[1] is radically different from the machines above described; its designer has sought to avoid every grinding action, and in that way to minimize the production of fine meal and pulp, and his success in this respect is beyond question. The essential parts of the roller mill are a large, slightly coned ring or annular plate, revolving about a vertical axis at the rate of 12½ revolutions per minute, and three conical rollers with fixed inclined axles, which are radial with respect to the central, vertical axis of the machine. The rollers are set upon the plate 120° apart. They rotate by reason of their frictional contact with the plate or with the ore that may be charged upon it. The coning angle of the rollers is calculated so that any two points which can come into contact —one being on the roller surface and the other on the plate—shall move with the same tangential velocity. This design, combined with slow speed, prevents dragging and consequently avoids all grinding action. The pressure exerted by the rollers in crushing is regulated by set-screws and rubber resistance buffers. Ore is charged and spread automatically at one point of the ring-plate, and passes successively beneath the three rollers, each one exerting a greater pressure than the preceding one. Upon passing each roller, the stuff is washed by a stream of regulated strength, which carries off into a circular launder the softer material which is already crushed, and leaves the coarse, harder parts of the ore upon the plate for the next pass. After the third and final pass the ore remaining on the table is mechanically scraped from it and put through a sizing drum in order to separate out insufficiently crushed material.

Experiments in reducing ore of 5 and 8 mm. sizes down to

[1] The Schranz Roller Mill; illustrated description and record of tests: Berg u. Huettenm. Zeitung, vol. xl., p. 357, 1881.

2.4 mm., both by stamps and by the mill, gave the following interesting results:

PROPORTIONAL QUANTITIES OF ORE OF DIFFERENT SIZES PRODUCED BY THE CRUSHING, AND EXPRESSED IN PERCENTAGES.

	3.2–2.4 mm.	2.4–1.6 mm.	1.6–0.9 mm.	0.9–0.5 mm.	0.5–0.2 mm.	Less than 0.2 mm.	First pulp settlings	Second pulp settlings	Finest basin settlings
Stamp battery.	....	4.68	15.15	16.96	24.08	16.72	5.87	3.88	12.66
Schranz mill..	6.95	21.07	26.27	16.92	15.31	7.21	2.23	1.71	1.83

The table shows that in using the roller mill 64.26 per cent. of the material was obtained in a "settling run" between sizes of 2.4 and 0.5 mm.; 15.31 per cent. consisted of fine meal and coarse pulp, and 12.98 per cent. was fine pulp; while the corresponding figures for the coarse stamping were 36.79, 24.08, and 39.13 per cent.

The capacity of the roller crushing mill, working on 3 to 8 mm. quartzose tailings from jigs, and reducing them as in the above table, is 3,200 pounds per hour, with a consumption of 25¾ gallons of water per minute for the crusher and 5¼ gallons for the sizing drum, amounting together to 31 gallons per minute, which is equivalent to 1,162 gallons per ton (2,000 pounds) of ore, or 965 gallons per ton for the crushing alone.

The slime resulting from this comminution is also washed without any preparatory reduction in volume. The sands and coarse meals from the machine yield, after washing, rich headings, poorer middlings, and tailings which, if still containing sufficient mineral to make the operation remunerative, are either crushed down in a second mill or accumulated and treated periodically on the first crusher. In the latter case, the rollers are set so as to press down upon the revolving ring with greater force than in the first crushing.

The durability of the wearing parts of this crushing mill is one of its noteworthy features. The crushing surfaces are of the toughest Bessemer steel. When seen by the writer at Laurenburg, they were perfectly even, and the wear did not amount to one-twentieth of an inch, though the machine had been in constant use for four months. One reason for this durability is that the angle between the surface of the revolving table and that of each roller at the point where the ore

grains first come into contact with both is very acute; hence the ore is seized at once, being favored in this by a slow speed, and does not slip and wear the surfaces, as is so often the case in ordinary rolls. Grains larger than 8 mm. (⅓ inch) have been treated in the mill, but not with profit; the coarse ore is not so readily seized by the rollers, and requires a special guard on the ring-plate to prevent its slipping off the latter.

The action of the Schranz mill is one of crushing by pressure only, as in a pair of common rolls. The other comminuting machines above described exert more or less of a grinding action, and the only way in which they can avoid the production of much pulp slime is to bring very little pressure to bear upon the ore; but by this method, from the very nature of the machines, a considerable portion of the ore must escape a proper reduction. Notwithstanding this acknowledged drawback, the various grinding mills have, as already mentioned, obtained widespread introduction, because they supply, though not perfectly, a long-felt want.

In some large works, grinding machines have been set on the third floor of the dressing mill; the ore passes through them and falls into a drum sieve, in which the oversized material is separated from the fine stuff, and then is fed directly into a smaller crusher below.[1]

In its crushing effect upon the ore, the Schranz roller mill appears, upon general considerations, to be more satisfactory than any of the other crushing mills. Its large capacity and favorable comminution, with the production of only seven per cent. of oversized grains (as seen in the last table), its slow wear, easy adjustment and small consumption of motive power (3.5 horse power), commend it as a comminuting machine, and would certainly lend interest to a direct competitive trial between it and the Heberle or the Bogardus mill.

The grinding mills can be employed in dry crushing, but for such work it is found that their capacity diminishes and their rate of wear increases. None of the present machines are adapted for the profitable reduction of a very finely mineralized ore to an impalpable pulp. When the grinding plates are set very close together, the rate of wear is very much increased. It is possible, however, that comminution by successive reductions may lead to favorable results. Treated by such a method, the ore would first be reduced in a coarse grinder and would

[1] Such a plant has been arranged at one of the Przibram mills.

then pass to a very fine grinding machine fitted with exceedingly hard and tough grinding surfaces, set perfectly true.[1]

HYDRAULIC CLASSIFICATION.

The object of hydraulic grading in quiet water or in a continuous, either rising or deep, horizontal current, is to classify the material subjected to such treatment into groups of particles, which, under like conditions, fall through the water together. In a perfectly free fall, the relative diameters of such equal-depositing particles bear an inverse ratio to their specific gravities, deducting the specific gravity of water from each. In practice, the friction along the sides of the classifier or the rolling of particles upon the bottom of the apparatus or on a bed of previously deposited ore, interfere more or less with grading according to the above simple law. The degree to which the action of any classifier approaches this law determines the efficiency or regularity of the classification.

All slimes resulting from comminution, or from the draining of coarse ore after jigging or any other dressing operation, are subjected to this grading process, not for the purpose of making final separations, but as a classification which is necessary before washing when the slime particles have a considerable range of size—from 0 to 1.5 or 2 mm.—as is commonly the case.[2] The products of such grading are two or three classes of *fine sand*, an equal number of *meal* classes, and from two to four sorts of *pulp*, or six to ten classes in all. The sands are usually charged upon fine sand jigs; the meals are treated on jigs or on round or percussion tables, and the pulp is generally washed on round tables.

The amount of water in the slimes is, as previously observed, dependent upon the comminuting apparatus that is used. Stamps crushing only to one millimetre necessarily produce a dilute slime. The extent to which classification shall be combined with concentration or reduction in volume, is a question for which most varied answers are found in practice. Slimes from the new types of grinding machines are not concentrated, but rather require, in the opinion of some, to be diluted. The

[1] Comparative tests for fine comminution between these machines and the slow but effective grinding pans of the West would be of great interest and value.

[2] Grading is even resorted to for slimes which have a much smaller range of size, as will be shown by example further on; in this respect there is a marked difference between Continental and American practice.

best consistency for the washing of any particular slime has not as yet, to the writer's knowledge, been determined. The more water that is used in wet slime dressing beyond a certain (still unfixed) limit, the greater will be the loss of fine mineral. The degree of dilution, however, must be such that the free movement of one ore particle is never materially interfered with by that of any other. The slime of clayey, slaty and calcareous ores, if reduced to as concentrated a condition as that in which slimes of quartzose or sandy ores are often washed, would have almost a semi-glutinous character, and the mineral particles would be maintained in suspension for a long time. Hence such slimes require a greater dilution and are treated in larger classifiers and on larger washing apparatus than the more favorable quartzose slimes, and their treatment is consequently more difficult and costly. Dilute slaty ore slime contains sometimes only 50 pounds of solid matter per cubic metre (2¼ per cent.), while more concentrated quartzose slimes are often found with 175 to 200 pounds of solid matter per cubic metre (8 to 9 per cent.).

The two extremes in which concentration and grading are simultaneously performed are found on the one hand in the system of depositing all the suspended matter of the slime in a series of long "strips," "runs" or troughs, and pits, suitably termed a *labyrinth*, and, on the other, in the development of continuous classification in pointed boxes and vats, or *spitzkasten*, to such an extent that the water overflowing from the last pointed vat contains no mineral that can be saved with profit. The first method, which is a survival of an old, primitive practice, still prevails in England, though abandoned in most continental mining regions. The second system has been introduced in several of the advanced mills—and is probably nowhere put to better practice than at Ems—but the majority of dressing works carry out a mixed system, sorting fine sands and meals in continuous classifiers, and leading the overflow from these into a labyrinth where the pulp settles.

The classification produced in pits or deep runs is fairly regular; the shallower the run, however, or the thinner the stream of slime flowing over the deposited sediment, the closer does the separation approach that made on buddles, but it follows a well defined law, and generally becomes very irregular and unsatisfactory. Such shallow current classification is now rarely seen in practice. The labor incurred in handling the

deposits of a large labyrinth, especially in the cold season, and the loss due to letting the settled material dry partially before it is mixed with water and washed, are very serious objections to the first mode of classifying and concentrating. The loss of float mineral from partial drying, familiar as this must be, seems frequently to be regarded with insufficient appreciation.[1] Its production is strikingly noticeable in every case where fine, dry or merely moist material is mixed with water and fed upon slime tables; fine mineral particles are then seen floating on the surface of the water, and being borne off by the current instead of caught upon the washing apparatus; in a few seconds they are carried into the waste launder or to large, final settling basins, where they form a scum, sometimes quite thick, on the water surface. This loss is inevitable, regardless of a high specific gravity of the mineral; to avoid its effects it should be a rule in wet slime dressing that when a finely comminuted ore has once been mixed with water, it must, if possible, never be completely deposited from it until the moment of the final separation of its constituents upon some washing apparatus. Governed by this rule, the abandonment of labyrinths even for the partial accomplishment of a general slime classification is a necessary result; as a classifier the labyrinth fills no place in a modern dressing mill which newer, continuous working apparatus cannot occupy to better advantage.

Continuous classification is now most commonly performed by means of the spitzlutte and spitzkasten. The syphon V trough, which is the old form of spitzlutte designed by Rittinger, is frequently met with in Austrian dressing works. It separates out of the ore slime a class of particles that are equal-falling in an upward current which is produced by the downward flow of the slime in one arm of the apparatus and its rise

[1] The writer, endeavoring on one occasion to make this source of loss very evident, treated some rich gold concentrates in a pan, and after collecting the fine native gold into a pure heading that was separated from the tailings of coarser pyrites and quartz, he exposed this gold to the air and then swept a sheet of water in gentle current over it. An exposure of a few moments on the pan had no effect in producing float mineral, but with a drying action of half a minute a very appreciable portion of the gold was carried over to the tailings as float: the operation on the headings was repeated quite a number of times, with the final result of washing away all the gold, which then continued to float indefinitely on the surface of the wash water. The adhesion of a thin envelope of air to each fine mineral particle, changing for the time its specific gravity, and the action of capillary repulsion observable upon close examination on the non-wetted surfaces of the mineral, are the productive causes of float mineral.

in the other; the heavy ore particles escape through an orifice in the bottom, while the light material overflows into the next spitzlutte. In newer forms of this classifier the same grading is performed in a rectangular box having a pointed bottom and a middle partition which reaches to within several inches of the apex.

A rising current of clear water is now always used in sorting *sand* classes, either with the spitzlutte or with a small spitzkasten classifier. The effect of the wash water in the last named apparatus is to produce the desired ascending slime current; in the spitzlutte it is to increase the upward current. The wash water also cleanses the sand and coarse meal from finer particles and renders it practicable to introduce in the bottom of the classifiers comparatively large orifices, which are easily kept clean. The rising current of clear water through the orifice is very easily produced by enclosing the mouth of the outlet with a small, closed box, to which water is admitted through a pipe under a head six to eight inches greater than the height of slime in the apparatus. The main portion of the wash water rises from the box through the orifice of the classifier, while the heavy ore particles fall down through the same opening and escape with the remainder of the water by a small outlet in the bottom of the box. To protect this lower outlet from wear it is lined with iron: the lining has the form of a small funnel to which is attached a long, light rod that passes up through the classifier; the rod affords a convenient means for raising the funnel to clean out accidental obstructions. The apex of the funnel frequently measures about 16 mm. ($\frac{5}{8}$-inch) in diameter for the first sand classifier, and 8 mm. ($\frac{5}{16}$-inch) for the last one of a set. Funnels of different diameters can easily be inserted when the quality of the slime changes.

The spitzlutte offers a small cross-section to the slime current; it is, therefore, particularly adapted to classifying a small volume of rather thick slime, or to sorting several classes of *sand* and *coarse meal* from a large volume of dilute slime. In the latter separation, for which the spitzlutte is often used, the slime passes through three or four of these classifiers at comparatively high velocity, and then enters a set of large pointed vats where the velocity of the current is very much reduced, and the fine meal and pulp particles are settled.

Theory has determined and experiments have verified the

velocities required in a rising current which shall maintain in suspension and carry off different mineral particles of given sizes. The tabulated results of such experiments[1] are used to ascertain what dimensions will be required for classifiers that are intended for the treatment of a given maximum volume of slime. To sort the first *fine sand* class[2] from the slime a velocity of six inches per second is frequently adopted; the succeeding members of a series of classifiers are designed with increasing dimensions, so as to produce a decreasing current velocity. The factor of diminishing velocity, which is usually between 0.35 and 0.50, depends on the quality of the slime and the number of classes that it is desired to produce. Though formulæ and experiments fix the size of the apparatus, yet for accurate grading the variability in the nature and quantity of the slimes renders it necessary to provide easy regulation of the sectional area of the current, the level of the overflow, the diameter of the discharge outlet and the supply of wash water. It is because of its adaptability for such regulations that preference is given to the spitzlutte rather than to the simpler, small spitzkasten classifiers. The latter, as commonly constructed, do not admit of changing the sectional area of the slime current. In order that they may perform a proper, unvarying classification, even with a slime supply of changeable volume, it is necessary to design the apparatus for a maximum supply, and then to resort to the unsatisfactory plan of diluting the slime with water by the amount which it lacks of the intended quantity.

At Schemnitz, Hungary, a very great importance is attached to an exact slime classification, because upon this feature the successful treatment of extremely fine, poor material is held to depend. The slime which is there treated has been described on page 46; it is a fine *pulp*, resulting from the stamping of an ore of auriferous quartz, fine galena, and copper pyrites to 0.4 mm. ($\frac{1}{60}$ inch) and less. The stamped material passes successively through two small amalgamation pans in

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[1] Rittinger, *Aufbereitungskunde*, pp. 192–203; also Transactions Am. Inst. of Mining Engineers, vol. 6., p. 415, 1877; or *Engineering and Mining Journal*, vol. xxiv., pp. 102 and 129, 1877.

[2] In ore classes sorted by their equal-falling properties, the size of each class is determined by that of its predominating constituent: in a sorted class of *fine sand*, for example, the bulk of the material may consist of grains of quartz of a true "fine sand" size (2 to 1 mm.), while the galena particles of the same class are not over 0.46 to 0.23 mm. in diameter.

which most of the gold amalgam and excess of mercury are retained. The slimes from the second pan are delivered upon a continuous working blanket table, which separates out nearly all the remaining gold amalgam and mercury; the tailings from this table enter a hydraulic classifier of twelve spitzlutten. The two coarsest classes of graded pulp from this apparatus are treated on separate Rittinger side-percussion tables; six middle classes go each to a Hungarian end-percussion table, and the four finest grades are washed on revolving tables.

The auriferous ore of Schemnitz is of very low grade, and the slime treatment of a large portion of it has to begin, in point of fineness, where the dressing works of many other localities have thought of ending. Yet the ore is profitably washed, and success, no doubt, is largely due to the thorough preliminary classification. A careful control of the quality of the slimes is always maintained in the mill; the workmen have deep test-glasses with which to take samples of the products issuing from the classifiers; for a practised eye the appearance and quantity of the sediment in the glass is a certain indication of the nature of the slime. It repeatedly happens that poor working on one of the slime washers is remedied, not by changing the tilt or speed of the table, but by re-establishing a normal condition of the particular grade of slime that is being treated. It seems evident that this course is the proper one, though its adoption is not common. If a table be set to treat a certain class of slime, but receive instead ore particles of a different grade, it will do poor work; this can be remedied by altering the running of the table to suit the new conditions, but then the adjoining tables which receive slimes from the same set of classifiers are in all probability either being overworked or are running partly empty, so that they also require a change of setting, while the mineral percentage in the overflow from the classifiers will at the same time often be found to have run too high. The more easily applied, radical and satisfactory remedy for irregularities in the slime is, therefore, to adjust the classifiers so as to preserve the quality of each slime class constant. The director of the Schemnitz mill, not content with the results heretofore obtained, proposes now, after a series of experiments, to render the spitzlutten still more susceptible to fine adjustment by substituting for each one of the present broad, syphon V troughs a number of narrow U-shaped iron pipes which shall collectively sort out a single slime class; the num-

ber of pipes thrown in or out of work will determine the sectional area of the current, and one of the pipes in a set can further be made capable of individual adjustment.

In the west of Europe the spitzlutte is not much used, even as a *sand* classifier. There the characteristic practice is to pass the slimes produced by fine crushing through a small drum screen, about two and one-half feet long and one and one-half foot in diameter ; this removes wood chips and oversized particles (which in the case of stamping, owe their presence to defects in the battery sieves); the cleaned slimes then flow to a *sand* and *coarse meal* classifier, consisting of four or six pointed boxes provided with rising currents of clear water. Each box may sort out a distinct class, or, as is more usual, especially for the finer grades, the products of two or more boxes are combined to form a single class, which is treated by itself. Such an arrangement is merely a contrivance for avoiding large, deep boxes. The smallest box for an apparatus classifying fifteen cubic feet per minute is ten inches wide, seven inches long, and seven and one-half inches deep at its apex, while the largest one measures twenty or twenty-two inches square and is fourteen inches deep. The quantity of wash water consumed by such a sand separator is 2.5 cubic feet per minute, so that it causes a dilution of $16\frac{2}{3}$ per cent. One classifier receives the slimes produced by fifteen to twenty-four stamps; the larger this number, the more uniform will be the nature of the collective product.

The overflow from the hydraulic *sand* classifier enters an extensive *fine meal* and *pulp* classifier, consisting of a series of large spitzkasten. For simplicity of construction, and to avoid very large, deep and expensive vats, all the spitzkasten of one set are made of the same size. The products from several vats are combined to form a single slime class, just as in the case of the smaller sand classifiers. A number of spitzkasten sorting in this way a single class form practically but one large vat having several pointed outlets in the bottom. In such a spitzkasten series there is a horizontal current of slightly decreasing velocity (because a portion of the slime is withdrawn at each succeeding outlet), but the effect of this is not sufficient to produce the required classification. To further influence the grading, therefore, the outlets at the bottom of the pointed vats are made of successively decreasing diameters, from three-fourths to one-fourth inch, and syphon discharges of various

lengths are used. The syphons consist of pipes rising from the pointed bottoms of the vats, and discharging at suitable levels into launders, which convey the different grades of slime to further treatment. The coarser the ore class that is to be sorted from the slime, the shorter is the syphon arm rising from the outlet. The effect of the syphon is to diminish the dynamic head under which the discharge takes place; this head averages three and one-half feet for the coarsest, and one and one-half foot for the finest classes. The use of the syphon reduces the net height required for the large vats down to about two to four feet, and it allows the introduction of comparatively large outlets which are not easily clogged.

The grading action of a set of equal-sized spitzkasten is one of equal-falling particles in a deep, slowly moving body of water. The coarsest *meals* settle in the first vat and are drawn off by a downward current through the discharge. This current is controlled by the size of the outlet and the amount of hydraulic pressure in the syphon. The regulation of the current is made according to the quantity of the settling ore class existing in the slime; acting continuously, the downward current of constant velocity exerts no influence on the character of the sorted material—it merely serves to promote the discharge. The more slowly depositing particles are carried from the first vat to a second and a third one, and so on, gradually passing over an extensive precipitating surface. The quantity of pulp settling in a single spitzkasten is often so small that the product could never be profitably worked alone, and hence, as above observed, the fine deposits of several large vats are collected to form a single slime class.

The force of the current entering a set of large spitzkasten is broken by a board set across the first vat, about eight inches from its head, and dipping six inches beneath the water surface. The spitzkasten are all open on top, and there seems to be very little trouble connected with their management. Experience has shown that the vats must be very substantially constructed, with timber bracing, and profitably with iron tie-rods; if lightly built, the vibrations in the mill soon cause leaking, and before very long the apparatus becomes worthless.

The classification of slimes at Ems has been referred to as a type of continuous grading without any handling of the products. Several slime classifications are carried out in different

departments of the works; they all involve the same principles, so that one may serve as an illustration. The slimes from sixty-eight stamps[1] (crushing to 1½ mm.) flow to five similar sand classifiers, each consisting of six small pointed boxes which together have 1 square metre (1.2 square yards) of precipitating surface. Each classifier produces two *sand* classes that are treated on four-sieve jigs. The overflows from the classifiers are raised by a centrifugal pump to a spitzkasten series of twenty-two pair of large vats with an aggregate of 82 square metres (98.4 square yards) of precipitating surface. The products of this fine classifier are four grades of meal and pulp, and an overflow which is discarded.[2]

The first product settles in the first three pairs of vats, with 7 square metres (8.4 square yards) precipitating surface; it is *coarse meal*, which is delivered to a double inward and outward-flow revolving table.

The second product is collected in five pairs of vats, with 19 square metres (22.8 square yards) of precipitating surface; it is fine, *second-class meal*, which is washed on a set of two outward-flow rotating tables.

The third product is *pulp*, from 24 square metres (28.8 square yards) of precipitating surface; this stuff is also treated on a set of two outward-flow rotating tables.

The fourth product, *fine pulp*, is obtained from 32 square metres (38.4 square yards) of precipitating surface, and carries all of the remaining mineral which can be profitably saved; it is washed on a double inward and outward-flow rotating table.

In one of the slime departments running on galena and blende ore the "middlings" from the second of each set of double washing tables are delivered to another large spitzkasten series, which grades the material into classes that are similar in size to those of the first separator, but different in mineral composition.

The system of slime treatment at Ems is manifestly charac-

[1] The figures in this case are taken from Bloemeke's description of the Ems Works, previously referred to: Berg u. Huettenm. Zeit., vol. xli., 1882.

[2] Formerly when a plain sieve battery was used for stamping instead of a stay-battery, a greater proportion of very fine galena was produced than at present, and some of this mineral was lost in the waste overflow from a set of sixteen pair of spitzkasten. An experiment at that time showed that in 10,200 cubic feet of waste slime, there were 9.2 cubic feet (equal to 760 pounds dry weight) of solid matter, carrying 22.8 pounds or 3 per cent. of lead. At present, with less dead stamping and a larger separator, the loss is undoubtedly less.

terized by the absence of any great degree of concentration during the process of classification;[1] the dilute products are washed on very large slime tables. It is of interest to note that in a branch of these same works a system has been developed which tends in a contrary direction to that just reviewed, and combines continuous classification with considerable concentration. In the *Mercur* Mill, at Ems, the dilute slimes, finer than 1 mm., pass through a classifier of four small pointed boxes, each with 0.5 square metre (0.6 square yard) precipitating surface; four products are formed, which go to Rittinger percussion tables. The overflow from the classifier enters a set of twenty-four spitzkasten, each with 1 square metre (1.2 square yard) precipitating surface. The deposit in this apparatus is run into a large sump, out of which the settlings are raised by a bucket dredge, to be charged into another classifier; this in turn grades the concentrated stuff into two classes, each of which is washed on a double inward and outward flow rotating table. An improvement of this system is embodied in the Geyer concentrator.[2] This apparatus consists of a series of pointed vats, each provided with self-regulating supply and discharge orifices. Slime is allowed to fill each vat, or set of vats, in turn, and have some time for settling before the outlets are opened to discharge the concentrated product. The automatic opening and closing of the orifices is effected by independent mechanism or by the flow of the slime itself. With such a system of highly concentrating spitzkasten, the precipitating surface used for treating a given quantity of slime would be much larger than that which the ordinary spitzkasten requires. On the other hand, smaller washing surfaces would suffice, and, as an important consideration, less water would be needed to carry the cleaned products off the smaller slime tables; hence settling tanks of reduced dimensions, and probably less loss of mineral, would be some of the advantages derived from the system, which seems particularly applicable to the dilute slimes of quartzose ores.

Slime Washing.

The important operation of washing graded slime classes

[1] It is reported that slime has been graded without any concentration whatever, its whole volume being treated on the washing machines.

[2] Geyer Concentrator, illustrated description: Berg u. Huettenm. Zeit., vol. xl., 1881.

for the purpose of separating the mineral from the gangue and producing final "concentrates," is performed on jigs, buddles, tables and sundry special machines. There still exists considerable difference of opinion regarding the relative merits of these apparatus for treating different kinds of material, and practice, therefore, presents analogous divergences. A few words upon each type of machine, followed by a brief consideration of some comparative tests, and of the effects produced by a change of apparatus in some of the mills, may serve to indicate the salient features of present practice.

Jigs.—The process of jigging has been described as the sorting of equal-falling particles in intermittent rising currents of water. To understand the separating action of the jig upon ore slime that has already been graded by hydraulic classifiers into classes of equal-falling particles, it is necessary to examine more closely into the laws of the free movement of bodies in water. These laws, formulated by *Sparre* and *Von dem Borne* thirty years ago, establish the following facts:

1. Particles of two different minerals whose diameters bear an inverse ratio to their specific gravities, the specific gravity of water being deducted from each, fall through quiet water or through a continuous rising current with a motion which at first is accelerated, but they soon attain an equal maximum and practically constant velocity.[1] During the first short period of varied motion, the smaller, specifically heavier mineral moves faster than the larger absolutely heavier body. The separation, by fine jigging, of "equal-falling" particles (using this term now in a limited sense to designate particles attaining the same constant maximum falling velocity), depends on the cumulative effects of the small gains made by the smaller mineral bodies in the beginning of each short fall to which they are exposed on the jig. This action, exerted upon equal-falling particles, resolves jigging virtually into a hydraulic sizing; when carried into effect on ungraded ore it will suffice to separate two minerals of widely different sizes, and this has been the basis of the process of hand jigging without preparatory screen sizing which is still used in Cornwall and Flintshire.[2] The hydraulic *sizing* of unclassified, or even graded,

[1] Von Sparre, *Zur Theorie der Separation*, p. 8. The period of accelerated motion is one second for particles 16 mm. ($\frac{5}{8}$ inch) in diameter, and 0.25 second for those 1 mm. ($\frac{1}{25}$ inch) in diameter.

[2] Experiments which have been worked into diagram form by *Althans* (Judges' Reports, Centennial Exhibition, 1876, Group I, p. 222) make it appear that galena

particles upon a jig is never as quick and economical an operation as the hydraulic *sorting* of screen-sized particles upon the same machine. Continental practice, therefore, restricts the former mode of treatment to ore which is too small to be economically sized by screening.

2. Of two particles which rise in an ascending current and attain the same constant velocity, the smaller (specifically heavier) body will move the faster during a small fraction of the first second. This action may seem an anomaly when it is considered that according to the first law the same smaller particle would also fall faster than the larger one through a weaker rising current, but the explanation of the relative motions is to be found in the inertia of the larger body: of two equal-falling grains of quartz and galena in a continuous current, the former body has twenty-three times the mass and absolute weight of the latter, and hence, from its greater inertia, it is slower to yield to the forces which are suddenly brought to bear upon it. This explanation also applies to the initial action of a descending current upon falling bodies, as developed by Rittinger[1] and stated below.

3. In a descending water current, the smaller of two bodies of the same specific gravity falls the faster for a fraction of the first second, and is then rapidly overtaken by the larger body. Of two equal-sized bodies moving in a downward flow of water, the specifically lighter will fall the faster if the current descend with a high velocity—*e. g.*, at the rate of 1 metre per second; but with a low current velocity the movement of the specifically heavier body is, in the beginning, the faster. In the latter case, however, the difference between the falling velocities of the specifically heavier and lighter bodies in the first short period of fall is not as great as the corresponding difference occurring when the bodies move through quiet water or a rising current. Under all circumstances, therefore, a descending current is a disadvantage in the jigging of *equal-sized* particles. Of two equal-falling bodies, grouped together as such in a slime classifier, the smaller (specifically heavier) body falls, in the beginning, faster than the larger body in a downward cur-

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grains could be separated by jigging from grains of quartz 38 times the diameter of the galena, if the jig stroke were reduced to 2.5 mm., and the duration of the stroke to $\frac{1}{10}$ of a second, though the ratio of the specific gravities of these grains, after deducting the specific gravity of water from each, is only as 4.06 : 1.

[1] Rittinger, *Aufbereitungskunde*, 1st Supplement, p. 33.

rent of water, and its advance over the slower moving body is greater than the similar advance made in quiet water. A descending current is therefore favorable to the jigging of *equal-falling* particles, and in this fact an explanation may no doubt be found why experiments made with slow return movements on fine jigs (that frequently treat "equal-falling" ore classes), have from time to time been reported as failures. A jig for separating the different constituents of a graded ore class works best with a regular piston motion; a quick return would promote a descending current and in so far be advantageous, but opposed to its introduction is the practical difficulty that it would tend to pack the ore upon the sieves.

The various equal-falling products of sand and coarse meal classifiers are often treated on four and five-sieve jigs; no rigid size limitation can be given for these products—the range is from 1.5 mm. down to about 0.5 mm. (the "mineral" being obviously much finer). The machines have been briefly described, under the general subject of jigging (p. 42). The maximum capacity of 8–10 tons per day, given for a meal jig, is only attained under the most favorable circumstances. In ordinary work, especially if the gangue contain any spathic iron, the machine will not treat over 6 tons in twenty-four hours. The stay-box commonly placed at the discharge end of a fine jig has frequently the shape of a small pyramidal funnel, and serves as a rough classifier; the fine tailing sands which pass through the neck of the funnel are washed on tables, while the overflow goes to waste or is used as wash water.

Buddles.—The hutches and pits in which the deposits formed in the extensive labyrinths of the old time mill were formerly hand-buddled by stirring and brooming, are now found only in isolated cases in any well ordered dressing works. The Cornish round buddle, in which hand brooming is replaced by the action of mechanical, revolving sweeps, is still the principal slime washer used in the English tin mining regions, but on the Continent it has made way for continuous working machines.

The action of the buddle upon the slimes is not very different to that produced in the Cornish classifying strips, the products from which are usually received by the buddle for further treatment. The separation must in this case, therefore, be a very poor one. The slime flows from the centre to the circumference of the buddle, or the reverse, depositing the

ore in an annular pit, while the overflow passes through a tail gate. The revolving sweeps act with most effect upon the coarser particles of depositing material, stirring them repeatedly from their positions and causing them to roll on toward the tail part of the ring; in this way a very imperfect and oft to be repeated sizing is effected. The deposit which accumulates in twenty-four hours to a depth of 14–20 inches in the buddle is divided, according to color, into "heads, fore-krazers, hind-krazers, and tailings;" the number of times that each of these products is re-washed depends on the richness of the material and the size of the mill. The ore, with 2½–3 per cent. of cassiterite, is dressed in the largest works up to 86–96 per cent. of "mineral." In order to effect this concentration the richest stuff that settles, after stamping, in the classifying strips is buddled three times and tossed in keeves three to six times—making six or nine washings in all, while the poorest product from the strips is subjected to at least thirty successive washing operations. An immense amount of labor is incurred. Of the mineral contained in the ore, 93–94 per cent. are saved in the best mills, and 3 pounds of tin oxide per ton (of 2,240 pounds dry weight) goes to waste in the tailings.[1] These tailings are concentrated in large catch-pits and washed over and over again by small independent "stream works," which are built all along the river banks from the mines to the sea, and are said to work with profit on slimes carrying only 1 pound of tin oxide per ton.

Results which are fully as good as those above given can be obtained on continuous machines with much less handling and washing of the ore. Considering the number of repeated washing operations, the percentage of saving, high as it is, cannot be regarded as satisfactory. Upon close examination it appears that much of the loss is due to float mineral which is produced by the frequent precipitation, drying and subsequent dilutions of the slime. Notwithstanding such unfavorable evidence, the round, deep buddle, which is the material exponent of the local axiom that "tin settles best on tin," is adhered to with characteristic pertinacity. Some experiments, made in the region, with continuous washers have been unsuccessful, either because of defects in the machines or in the mode of running them, and such failures have been assigned as reasons for retaining the buddle. A depressed tin market

[1] The loss not unfrequently rises to 6 pounds of *tin* (cassiterite) per ton.

and the impoverishment of many of the mines have not encouraged capital to invest in new machines; and, moreover, the introduction of any mechanical washers that require an efficient transmission of power and a good protecting covering would necessitate an entire reconstruction of most of the mill structures, for their present condition is most dilapidated. Any system of dressing by which it is possible for subordinates in the regular mills to be pecuniarily interested in such enterprises as the "stream works," will always find some advocates whose motives cannot bear close inspection. At West Seton, Cornwall, a Frue vanner has been used experimentally to wash the tin ore; it is reported that the concentration was considered quite satisfactory, and it was, of course, much more quickly effected than would have been possible by the old methods, but the cost of an expensive plant prevented the adoption of the machine.

Stationary Plane Tables.—In a class of graded, equal-falling particles, the diameters of the different minerals are, in practice, approximately in inverse proportion to their specific gravities after deducting the specific gravity of water from each. The separation of the different minerals of such a graded class is effected on a stationary inclined table by spreading a thin sheet of the slime upon it and allowing the same to flow slowly down the incline; the different mineral particles which roll a short way down the table surface and are then deposited from the slime—the latter flowing with any given velocity—have diameters that are inversely proportional to the squares of their specific gravities after deducting the specific gravity of water from each.[1] A class of equal-falling particles, whose diameters differ inversely as their specific gravities (each minus 1), are therefore separated into two or more groups with diameters inversely proportional to the squares of their specific gravities (each minus 1), and consequently the practical effect of the operation is a sizing. The fine, specifically heavy particles settle on the table, while the coarse, specifically lighter ones are carried off.

The importance ascribed by Rittinger to an increasing force and velocity of the thin sheet of flowing water at increasing heights above the table surface does not seem to be generally conceded. Investigation has shown that by far the most important element which interferes with efficient sizing on

[1] Sparre, *Zur Theorie der Separation*, p. 28.

the washing table is an undue rapidity of the water current; the mineral particles, caught by a comparatively strong current and held partly in suspension, move by *sliding* instead of *rolling*, and are quickly carried into the tailings. To prevent this action as far as possible, the slimes are allowed to flow over the tables at very low velocity, and a portion of the gangue deposits with the mineral; then, by brooming, or by the action of a current of clear water, or both, a second sizing is performed. This cleaning or "washing" process is repeated one or more times so as to produce headings of sufficient purity; the middle products, washed off the table in the successive operations, are re-treated, while the tailings which flow over the table in the first separation go to waste.

The old intermittent table, worked with brooms by manual labor, is still found in some of the mountain districts where labor is very cheap, but even in those regions it is gradually being replaced by continuous apparatus.

An automatic plane table, known as a "frame," is used in England for concentrating very dilute slime. The slime is allowed to flow upon the inclined table, and all the overflow passes at once into a foot or tail launder. Wash water is charged intermittently upon the depositing material from two troughs, set one at the head and the other over the middle of the table; clapboards are opened at the same time and the washings are collected in two launders as heads and middlings of very low grade. The intermittent delivery of the wash water and the opening of the clapboards is very simply effected by a small continuous stream of water which fills little buckets that are attached by levers to the wash water troughs; when one bucket is full, its weight tips the nearly balanced trough, which is thereby discharged, as is also the bucket itself; immediately a second bucket begins to fill, and soon brings the trough back to its first position, in which it is replenished. The clapboards are moved by rods connecting them with the troughs. One boy attends to the proper working of two hundred automatic frames.

Tables spread with coarse linen cloths are occasionally used for the separation of gold from copper or lead minerals, but they too are being replaced by continuous machines in the few large works where such separations are made.

Continuous Round Tables.—The most commonly applied form of continuous slime washer for treating sorted fine meal

and pulp is the round table. Its separating action is the same as that of the inclined plane table, but as the velocities of the slime and wash water currents vary in flowing over the round tables, the character of the depositing material is additionally influenced by the law that the diameters of two particles of the same specific gravity deposited in shallow currents of different velocities are directly proportional to the fourth roots of those velocities.[1] Hence the slower the flow of water the finer will be the deposited mineral of any one species.

The ordinary revolving outward-flow, or outward discharge, table consists of a flat cone with a dip of approximately 5°–10° (from 1:12 to 1:6), and a diameter of 3.5 to 5.5 m. (11.5–23 ft.). The table surface is made of pine or beech, or preferably, of narrow maple boards, secured to an iron frame of radial arms, 8–16 in number, set into a vertical rotating iron axle. At the centre, or head, of the table is a small fixed apron upon which the sorted ore slime is continuously delivered from a launder. The slime is allowed to spread over fully one-third of this circular apron, and falls from its perimeter in a thin sheet upon the revolving table, with but very slight initial velocity. The crude slime covers at one time a sector of about two-fifths of the table, and the overflow from the lower edge of this sector is received in the corresponding segment of a fixed circular launder, which communicates with the waste tailing system. The deposit left by the slime upon the table is moved around by the rotation of the machine, and brought under the action of a current of water which is delivered, like the crude slime, to a part of the fixed apron, and thence spreads over the precipitated material. That portion of the deposit which the water washes off the table into a second section of the launder, reaching around one-third to two-fifths of the table circumference, forms a poor middle product; a rich middle product (sometimes constituting a second quality of headings) is frequently carried by the wash water to the lower part of the table, where, owing to the decreasing force of the current, which is there spread over a large area, the material is again deposited. A special water pipe or small trough suspended over that portion of the table where these rich middlings begin to settle delivers wash water upon the machine to carry this product into its proper segment of the circular receiving launder. Finally, nothing but the mineral headings remain where

[1] Sparre, *Zur Theorie der Separation*, p. 28.

the slime was originally charged; this product is removed by strong water jets or by brushes aided with a little water. The jets alone consume much water (which always implies large settling tanks and increased difficulty in preventing losses), and they leave an impalpable, slimy coating on the table, necessitating an occasional cleansing of the washing surface with pumice. The brush which wears best and requires very little motive power is made of bristles set on a long rotating radial axle, which is placed parallel with the incline of the table.

It is often an advantage to protect from the action of any wash water the richest headings which deposit on the table: hence in a few mills the wash water, instead of being delivered upon the rotating surface from a fixed central apron, issues from holes in a spiral pipe which is suspended above the table, and so curved that the best mineral headings are left untouched by the clear water.[1]

The proportion of intermediate products obtained on the round table depends obviously upon the nature of the slime; it is therefore necessary that the positions of the partition boards which are set in the circular receiving launder, dividing it into several segments, can be easily altered.

The speed of the tables varies very much in different mills; in washing a very dilute slime at Clausthal large tables are run at a speed of one revolution in 3–4½ minutes. Commonly, however, they revolve at a quicker rate, and especially when treating more concentrated slimes, which are delivered upon the tables faster than dilute slimes. For example, to wash a meal slime carrying 8–9 per cent. of solid matter, a speed of 2–2½ revolutions per minute is not unusual. High speed increases the capacity of the table; the limit in this direction is reached when tailings from the washer cannot be kept to a low grade.

One to one and one-half horse-power is required for a table which is worked with the old fashioned reciprocating scrubbing brushes, but when the cylindrical brush, above described, is

[1] This principle is carried still farther, with great advantage, in the *Evan's* patent apron with spiral perimeter, which is in common use throughout the Lake Superior copper region. The deposited headings at any one point on the round table are protected from the flow of fresh slime (as well as of wash water) by their passing at once beneath the apron. A rapid charging of the slimes is made possible without loss in the headings, so that a large table washing a very fine slime of native copper with a silicious gangue treats 30 tons in twenty-four hours, producing at the same time tailings carrying only 0.75% copper.

employed, one-half to three-fourths of a horse-power suffices for the apparatus.

A very fair washing capacity for a table is 15 cu. m. (22.5 tons, dry weight) of coarse meal, or 4 cu. m. (6 tons) of fine pulp in a day of twenty-two working hours. These figures apply to an easy dressing ore (for example, to galena and copper pyrites in a quartz gangue), and vary widely with the character of the slime. A very fine or rich slime is delivered upon the table very slowly, so that the mineral may find ample time to settle, and comparatively little wash water is used in treating it to avoid high losses in mineral—in a word, a rich or very fine slime requires slower washing than a poor or coarse one, and hence reduces the capacity of the apparatus.

In constructing the table surface, the grain of the wood is set at right angles to the direction of the slime flow; the wood swells and raises small ridges, which act favorably in hindering the sliding motion of the fine mineral. From time to time the table surface is purposely roughened with a sharp pronged rake.

A round table with inward flow, known as a central discharge table, is not an uncommon form of slime washer. It is shaped like a flat funnel, and the general proportions and action of the apparatus are similar to those of the outward discharge table. The slime is fed upon the revolving table by overflowing from a fixed, circular supply launder which extends around one-third or more of the outer perimeter of the apparatus. A second launder joins the first and completes the ring about the table; wash water overflows from it on to the table and produces middlings, just as in the case of the outward-flow apparatus.

The true merits of each of these forms of slime washers are recognized in a number of dressing works, where both are used in combination. On the inward-flow table, the slime is spread in the very beginning upon a large area, so that the force of the current is comparatively small and the mineral has a large precipitating surface; these conditions favor the deposition of headings. As the slime flows down the table, the area becomes contracted, and the force of the current increases, and therefore none but the coarsest of the middlings can be deposited; this condition is unfavorable to the production of barren tailings. The outward-flow table, on the other hand, offers a small area for the precipitation of headings, and a very large one, with a current of diminishing strength, for the deposition

of fine middlings; it is therefore especially suited for producing barren tailings, and not so good for precipitating the headings. When used in combination, a central discharge table is mounted above an outward discharge table on a common vertical shaft, and the tailings from the first are immediately re-washed on the second. The result of such treatment is decidedly better than two successive washings on tables of the same form.

In the disposition just described the inward-flow table has to treat more slime than the lower outward-flow table, and for this it is peculiarly well suited. The comparatively strong and rapid current that so effectually carries off the tailings promotes a large capacity which is one of the characteristics of the inward-flow table. Of several such tables, 16 feet in diameter, washing the finest kind of pulp at Clausthal, each one treats 6¼ tons (of 2,000 pounds, dry weight) per day of twenty-two working hours. Under the same conditions, the capacity of an outward-flow table was found to be a little over two-thirds as great.

The products from a typical treatment of fine argentiferous galena and blende slime, coming from a hydraulic classifier and washed upon a set of inward and outward-flow tables, are as follows (subject, of course, to many modifications):

Upon the inward-flow, upper, table:

1. *Headings*, galena with 35–50 per cent. lead, according to the degree of purification to which washing is carried; dried and sold.

2. *Middlings*, galena, coarse blende and gangue; returned to the same table, or, preferably, washed once or twice on a separate table—or sometimes on a meal jig and then on a table—for galena and blende.

3. *Tailings*, blende with gangue and fine galena; delivered directly on to the lower table.

Upon the outward-flow, lower, table:

4. *Lower Headings*, a marketable second quality galena concentrate; or, more often, an enriched product which is re-washed—sometimes on the upper table—yielding a rich galena (sold), and a middle product, mainly of blende, that is once more treated for high grade marketable zinc sulphide.

5. *Lower Middlings*, blende and gangue; usually washed with the middlings of the upper table, though occasionally returned to the hydraulic *meal* and *pulp* classifier.

6. *Lower Tailings*, barren gangue; discarded.

At one of the mills a pair of tables—the upper one 3.15 m. (10′ 6″) in diameter and the lower one with a diameter of 5.4 m. (17′ 8″)—treat in a day of twenty-two hours 72 cu. m. (96 cu. yds.) of slime which carries 6_3 tons of very fine pulp, or approximately 8 per cent. of solid matter. The headings of the central discharge table carry 45 per cent. of lead, and those from the lower, outward discharge table 30 per cent. of lead, and the discarded tailings run 0.40 per cent. of lead.

A stationary, continuous working, outward-flow table, designed by Linkenbach, the superintendent of one of the Ems mills, has been in use for several years. The table itself is fixed, but both the supply and receiving launders revolve. The washing is, therefore, of the same kind as upon a revolving table, but the advantages gained by the new design are cheaper construction and the possibility of using very large tables requiring but small motive force. To economize space in using such large washing surfaces, and to farther cheapen their cost, several tables have been superimposed, producing the apparatus [1] shown in section in Fig. 3. The diameters of the tables A, A′ and A″ are respectively 6, 6½ and 7 m. (approximately 9½, 21 and 22¾ feet). Of three grades of classified slime that are washed at the same time on the apparatus, the coarsest grade is treated upon the upper table, and the medium and finest classes respectively upon the middle and lower tables. In the vertical axis of the apparatus is a revolving cast-iron pipe, B; connected to, and revolving with, it are the vertical adjustable gates, C, C′ and C″, the perforated wash water pipes D, D′ and D″, and the inclined perforated pipes E, E′ and E″, which furnish water jets for carrying the headings off the tables. The receiving launders, F and F′, are stationary, but the lowest one, F″, is supported on small wheels and revolves. The bottom of each of the upper launders, F and F′, consists of a ring of flat funnels, the necks of which are 600 mm. (24 inches) apart; they deliver into a number of nearly vertical pipes, P and P′. The slimes are delivered upon three distributing aprons by the supply pipes G, G′ and G″; then, in one revolution of the axle, they are discharged through the revolving gates and successively cover the whole washing surface. The clear water is supplied through the pipe O, and passes into the hollow axle

[1] Illustrated description of the new Linkenbach table: *Wochenshrift des Vereins Deutscher Ingenieure*, 1882, No. 9.

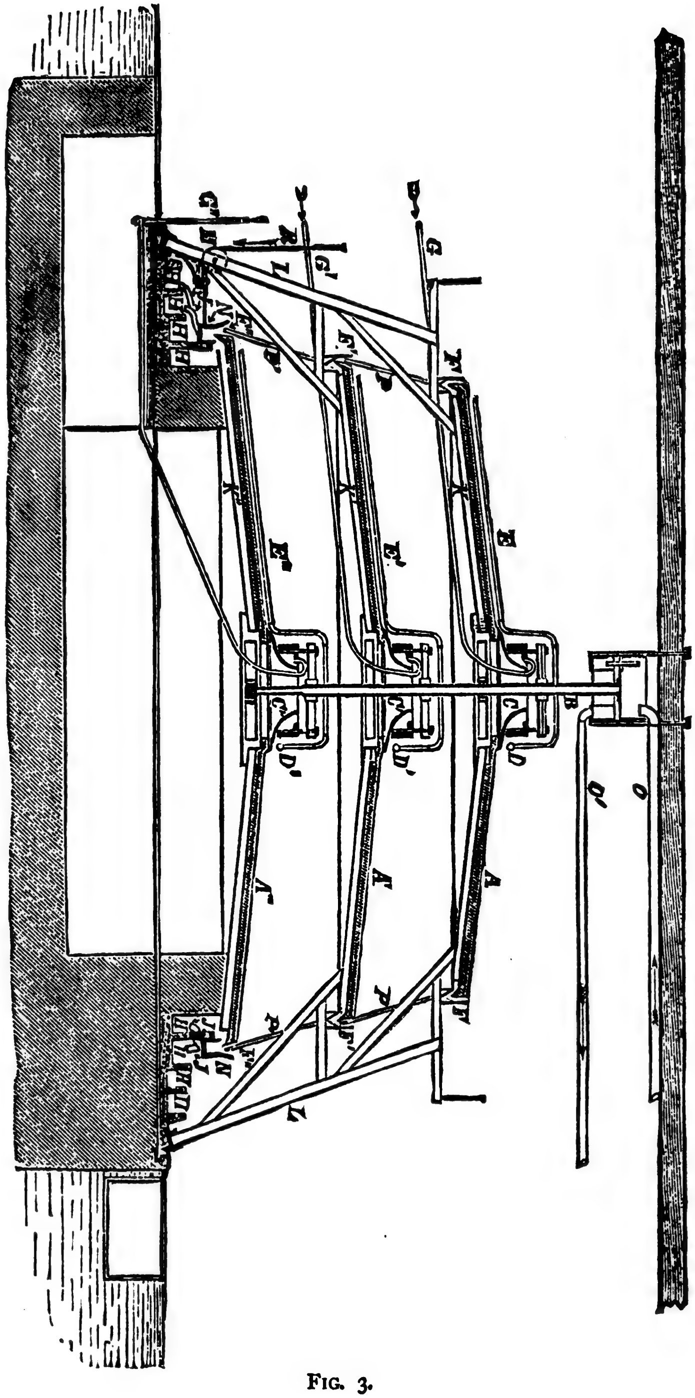

FIG. 3.

and thence to the revolving wash water pipes; an overflow is provided for in the pipe O′. Every part of the launders, F and F′, receives in succession during one revolution the tailings, middlings and heads of the upper tables, and discharges these directly through the pipes P and P′ into the revolving launder F″. The latter is divided by partition boards into segmental sections of adjustable lengths; it revolves with the same angular velocity as the other movable parts of the apparatus, and therefore each section can be made to collect a separate product which it receives through the pipes, P′, from above, or directly from the washing surface, A″. Each part or section of the revolving launder is supplied with a spout, J, through which its contents are discharged into one of the four circular troughs, H, that connect with an extensive system of settling tanks.

The tables, A, A′ and A,″ are made of cement, 80 mm. (3¼ inches) thick, set upon thin iron plates which are secured to iron frames, K, K′ and K″, each of which has eight radial arms. The lowest frame, K″, is bedded on the foundation masonry, while the upper frames are sustained by eight iron supports, L. The revolving motion is obtained from the belt R, which passes under the pulley M, and then around a ring of angle iron, N, which forms the outer periphery of the revolving launder, F″; from this launder the motion is transmitted to the central axle by means of the pipes, E″ and D″, of the lowest tables.

Single tables of the Linkenbach system have been built as large as 8 m. (26 ft.) in diameter. The apparatus at Ems concentrates the headings to 38 per cent. lead, but has been worked up to 50 per cent. lead, and even, for the sake of demonstration, to 65 per cent. lead. A high concentration brings a greater proportion of mineral into the middlings and tailings, but with the first named lower concentration it is claimed that the tailings are unusually clean. The capacity of a single 8-meter table is 14½ tons (of 2,000 pounds, dry weight) of fine meal and pulp in 22 hours. For washing coarser slime the capacity is correspondingly greater.

End-Percussion Tables.—A large number of mills on the Continent still use the old intermittent end-bump table for washing graded classes of *meal* and *pulp*. An apparatus of this type consists of a plane table, 4′ wide and 8′–12′ long, hung by four rods to as many iron standards, which form part of a fixed

frame. The table is shoved forward by the action of a rotating cam-shaft, and tends to return by gravity to its first position; it is checked in this return movement by striking against a cushion, and the shock is transmitted to the slime which flows in a thin stream over the table, and to the deposit which accumulates upon it.

The inclination of the table surface (2°–5°), and the amplitude of movement are subject to regulation, both diminishing with the size of the ore. A stroke of 5″–6″ is used for coarse meals, and 2″ for pulp when an elastic cushion (usually in the form of a wooden beam, 3″–4″ thick) is employed. A cushion of this kind is generally preferred to a rigid resistance block. The cam-shaft imparts 16–24 displacements per minute to the table, and the cushion causes three rebounds between the regular strokes, so that the ore receives 64–96 shocks per minute in a direction opposed to the flow of the slime. In cases where a rigid block is used, the stroke varies from 2″–0.5″, and speeds of 150 strokes per minute are sometimes employed.

When the apparatus is properly worked, each shock tends to smooth the surface of the accumulating mineral and to compact the deposit, but its principal effect is to retard, or arrest, for a moment the flow of slime, imparting to it even a slight retrograde motion. This sudden check of the current brings to rest all the ore particles, many of which had assumed a sliding motion, and with the beginning of the renewed flow they commence to move forward by rolling, which motion (as observed on page 73) is the one best suited for slime sizing. In so far, therefore, the intermittent flow is an advantage, which, however, is partly counteracted by the direct unfavorable effect of the end-bump upon the separation of the ore particles themselves: in an equal-falling ore class the grains of gangue, which are much larger than those of the mineral, have the greater inertia and are impelled farther than those in the backward movement given by the shock, so that they are continually obliged to overhaul the mineral particles during the periods of flow.

The slimes treated are almost all of very low grade. The tables are always so worked that in the first separation, barren tailings are produced. The deposit upon the table, after accumulating to a depth of 6″–8″, is divided into heads, middlings and tailings, and each product (with the exception of the first tailings) is re-treated a number of times, according to its richness

and the degree of concentration desired. The first middlings are sometimes subjected to finer comminution before being re-treated. The slimes which flow off the tables during the re-treatment of precipitates settle in runs and are worked over. The number of re-treatments varies from three to a dozen. At Freiberg, Saxony, the practice in one of the old dressing mills is to produce seven slime classes by the hydraulic grading of stamped quartzose ore; the coarsest class, treated on a four-sieve jig, yields pure galena, marketable mispickel (arsenopyrite, specific gravity, 6.3), impure pyrites mixed with blende and building sand. The six finer classes are separately washed on end-bump tables, and each is subjected to eight successive washing operations, in the course of which the following final products are obtained in the order here named: Barren tailings (discarded), blende mixed with some pyrites, pyrites, mispickel, medium galena and rich galena (all dried and sold).

The tables are conveniently arranged side by side in sets of three, with a space of 10–15 feet between the different sets, so as to allow room for the accommodation of intermediate products. Each set of tables works up a certain amount of stuff from the condition of raw slime to final product. The time required to form a deposit 8″ thick upon a table is about two hours for fine ore sands, and nine to twelve hours for pulp; in re-washing precipitates (which are always comparatively fine), six to nine hours are consumed in filling one table. During the removal of the deposited material the tables are frequently allowed to continue in operation. One table can treat at most 30 tons of coarse meal or 5–6 tons of pulp per day of 22 working hours, but, as the headings from this washing are in nearly all cases twice re-treated, and at a much slower rate than the original material, the average daily capacity of all the tables of a mill cannot be counted at more than 8–10 tons of ore-slime for each table in treating coarse meal, and 1½–2 tons in washing pulp.

There is obviously much manual labor connected with this system of washing, and a certain loss of float mineral is also incurred. No wash water, however, is used in the process, and the slimes prepared for re-treatment are generally very concentrated.

The importance of a uniform quality of slime, and the impossibility of obtaining such by diluting precipitates with water (even with the use of various mixing tubs and mechani-

cal stirrers), is shown by an examination of sections cut through the deposit upon an end-bump table. The deposit exhibits a decidedly banded structure; comparatively coarse grains are superimposed in layers upon finer ones, which frequently have a different shade of color, and thin, sharply defined seams of rich mineral are often found extending far down toward the tail end of the table, instead of being deposited as headings with the rest of the material of its kind.

Side-Percussion Tables.—Rittinger's well known continuous side-percussion table is much used for treating sorted *meal* slimes in Austria and also in Belgium, where its construction has been highly perfected. In Germany it is by no means uncommon, though introduced there to less extent than in the above named countries.[1] It is essentially an inclined plane table, suspended from four fixed standards; it is subjected to lateral displacement by a revolving cam-shaft, and then, by means of a spring, is driven back to its original position and strikes against a resistance block; it receives thereby a side shock which affects the thin sheet of slime that flows over the table surface.

The washing surface is always divided by a longitudinal partition board into two equal parts (each 2 m. long by 0.9–1.50 m. wide), so that every table is virtually double. The inclination of the table (3°–6°), and the amount of displacement (⅜–2 inches), are subject to adjustment, the smaller figures applying in each case to the mode of setting for working pulp.

Slime is charged at the head of the apparatus—in the upper right or left-hand corner—in a thin sheet 8–12 inches wide. It flows down the table, and in its course the ore particles are driven toward one side by the lateral shocks, and describe, as resultant, a curved, diagonal path. Each individual shock has more effect upon the absolutely heavier particles of gangue than on those of the mineral, but the latter, moving more slowly down the table, are subjected to a greater number of impulses, and therefore diverge more than the particles of gangue from the straight line of flow. The different products of the slime thus follow different lines on the table, and flow into separate receivers at the lower end of the apparatus.

[1] In Sweden, the Rittinger table has been introduced, notably in large dressing works erected near Falun for the treatment of a poor ore of finely disseminated copper pyrites, carrying a little gold. Heberle, *Das Neue Aufbereitungswerk zu Falun*, Berg. u. Huettenm. Zeitung, 1872.

Wash water is delivered in a wide, thin stream upon the head of the table, alongside of the slime; its quantity increases toward the heading or percussion side of the table. It promotes the downward course of material that is driven into its line of flow, acting with more effect on the gangue and intermediate products than on the fine head mineral, and it serves to prevent the deposition of any material upon the table. The delivery of the wash water in a thin sheet (and not in jets, as to a certain extent it is used on round tables), necessitates a large water consumption in carrying the headings off the table, and this unavoidably involves the production of a large class of middlings, containing more or less head mineral.[1]

The washing action of the side-percussion table is identical with that of a stationary plane table. The shock has not, as in the case of the end-bump table, a beneficial effect upon the slime flow, but is merely a device for effecting a continuous delivery of each product. It does, however, influence the quality of the products—not as they are sized by the washing, but as discharged from the table—for variations in the speed, or, more especially, in the throw, can bring together into the same line of flow particles that had been previously separated by washing. For any given slime there is a certain proper throw of the table which is found by experiment; the effect of increasing it is to bring middlings into the head mineral; by diminishing it, a very pure heading is produced, but a larger proportion of head mineral is carried off with the middlings. This control of the quality of the headings by means of the percussion is often utilized: a certain proportion of the headings produced by washing is frequently allowed to flow off the table with the middlings; this portion will naturally contain the poorer constituents of the headings, and therefore the final head product of the table is raised to a correspondingly higher grade.

The tables are usually run to produce highly concentrated headings, which constitute a very small class of the products, and one or two large classes of middlings and barren tailings. In the treatment of an ore carrying several minerals and rock gangue four products are not unusual, and even five are

[1] In home practice, the capacity of the Rittinger machine has been considerably increased by carrying the headings off the table by a line of sharp water jets, and the quality of the operations has been favorably influenced by an addition of two feet to the usual length of the table.

formed in washing the Schemnitz ore slime, which has already been described (pages 46 and 62). In that case the table yields:

1. *Extra Heads*, a very small quantity of gold amalgam with galena; the two products are separated by hand washing.

2. *Heads*, rich galena concentrate; settled in long runs, which yield headings (sold), middlings (coarser than the headings—washed on end-bump tables), tailings (ditto), overflow (discarded).

3. *First Middlings*, copper pyrites, settled in runs; products treated as in (2).

4. *Second Middlings*, poor slime; passed through a small spitzkasten yielding an overflow (discarded) and a precipitate (re-treated on same table).

5. *Tailings*, barren slime; discarded.

It sometimes happens that the tailings from the table carry fine, scaly head mineral, at the same time that the adjoining middlings are wholly free from it. This was the case in dressing works at Bleiberg, in Eastern Belgium, near Aix-la-Chapelle. To prevent mineral loss from this source, a thin cleat is set diagonally across the flow of the tailings, from the washing toward the slime-feeding side of the table; the water above the cleat is stayed, so that the mineral has time to settle; as soon as it touches the table surface it is affected by the lateral shock, and gradually driven over to the heading side of the table.

Experience has shown that good results are obtainable only with an exceedingly uniform quality of slime, or else by constant attention and frequent changes in regulating the machine.

Many improvements have been made in the side-percussion table, with a view to securing better separations, greater durability and a larger capacity. Very smooth washing-surfaces of fine grained wood, of wood covered with rubber cloth, linen and zinc, of glass, cement, slate, marble and planed cast and wrought iron have been tried at different mills. Surfaces of iron or of marble and slate are now generally adopted as the best—*i.e.*, they are adapted for good work and are at the same time very durable. At some works, comparative experiments have led to the preference of iron over marble for the washing surface,[1] but to any one closely observing the separation on

[1] See results of tests at Przibram; Transactions of the Am. Inst. of Mining Eng's, 1879, Vol. IX., p. 443.

an iron surface that has been in operation for a long time, the impairing influence of roughness that is due to rusting must be apparent. Iron, after having been much used in Belgian works, has yielded there in a number of cases to slate and marble.

Great durability has been obtained by the use of firm masonry foundations, and by constructing the framework of iron, or, more recently, of Bessemer steel, with which it can be made both lighter and sometimes cheaper than with iron. To avoid a tendency of the frames to creep sidewise on their bedding, it is very good practice to neutralize the effect of the lateral shock by securing the framework of at least two double tables to a single foundation sill, and then, in working, to push the tables in opposite directions by cams set upon a common shaft.[1] The wear of the cams has been diminished by introducing rotating tappets, on the principle of the California stamp.

The movable tongues which are commonly used at the lower end of the table to lead the separated products into different sections of a fixed receiving launder, have been done away with at several mills. Each tongue, in the ordinary arrangement, is secured by a thumb-screw to the table, and can be set to correspond with the line of division of two distinct products. When worn, however, the tongues are apt to shift their positions with the continual shocks, and they always shorten the active separating surface of the table. Movable receiving trays of zinc or sheet-iron have been substituted. These receivers are 12–18 inches long; they set within the fixed transverse foot launder of the machine, along which they can be slid at pleasure, and are supported by small flanges upon the launder sides. The bottom of each tray inclines slightly, and in a direction at right angles to the length of the table; for every product there is a separate tray, each one being set with its upper end on the line of division between two products. The separated slimes flow into the tray-receivers and are turned at right angles and led into the various launder sections. Two or more receivers are often partially superimposed; they answer the same purpose as movable section partitions in the launder; with the employ-

[1] This mode of setting has been introduced at Dam, near Antwerp, in works treating a finely mineralized Sardinian ore of galena, blende, and copper and iron pyrites.

ment of these, however, each change in the position of a partition piece would necessitate a cleaning out of all precipitate that might have accumulated in the launder near that point, but this trouble is avoided by using the sliding trays.

The capacity of the table and the consumption of wash water depend, as in other washing tables, upon the nature of the slimes—fine and rich material always requiring slower feed than stuff that is coarse or poor. The average capacity as given by Rittinger for one of the early tables with two compartments was 2½–3 tons of graded *meal* in 24 hours—not including the final concentration of the middlings. The speed was 90–100 shocks per minute for ordinary meal slime, and 120–140 shocks for finer material. The small capacity was due partly to the fact that only two-thirds of the washing surface were covered with slime, the remainder carrying nothing but wash water, and it could be partly ascribed to the character of the slime flow: the latter, for any given dip of the table, attains almost at the start a maximum, practically constant velocity, which is neither interrupted as on the end-bump table, nor gradually diminished in force as on the outward-flow round table; the slime supply must, therefore, be small and very carefully regulated to prevent the mineral particles from *sliding* off the table with the current. But the principal trouble was recognized not so much in slow washing as in the time required to move the different products by percussion into separate lines of flow. Hence, as might be expected, a greater capacity was attained with higher speeds—200–300 shocks per minute, and a throw of $\frac{5}{16}$–½ inch. Working under such conditions and using 1–1⅛ horse-power, a table with two compartments can treat 6 cu. m. (9 tons) of ordinary meal or 15 cu. m. (22½ tons) of coarse meal per day of twenty-two hours, The concentrate is, with rare exceptions, a high grade final product. In separating galena, blende and rock gangue, an almost pure galena heading can be obtained, running only 1–2 per cent. of blende, and at the same time the tailings can be of so low a grade as to be discarded, but the quantity of headings produced by such a separation is very small—much of the lead remaining with the zinc middlings, which require re-treatment.

Various Patented Slime-Washers.—In addition to slime jigs, round tables and percussion tables, a number of other machines have been designed for slime washing, but these devices,

most of which are patented, have been introduced to but very limited extent in practice. An apparatus consisting of an endless canvas or blanket belt moving across the line of the slime-flow is quite an old form of continuous plane washer, which is sometimes used instead of the old stationary blanket tables for washing gold or amalgam, or argentiferous galena slimes. The nap of the cloth prevents, to a certain extent, the loss of fine mineral particles, which are apt to float off with the current. The belt is inclined in the direction of its breadth, and travels over two revolving drums. The ore-slime is delivered upon the belt at one end of the washing surface, and the poorest stuff flows down at once into a tail launder. The depositing material is moved sidewise by the progressive advance of the belt, and is subjected to the action of a broad current of clear water, which carries off a certain portion of the material into a launder for middlings; the headings remain on the belt until it passes below one of the end drums, and then they are removed by fine sharp sprays of water. A variety of machines of this type have been designed, differing only in points of detail that are intended to diminish their cost or increase their durability and efficiency. For the treatment of large quantities of slime such machines are not used; many dressing works, it seems, have become prejudiced against them on account of the practical difficulty which for a long time was encountered in securing cheap and durable belts. Another more permanent objection to these belt machines with transverse travel lies in the short length of the washing surface, which is limited by the breadth of the belt.

The Frue vanner, so well known as a slime washer in American practice, has only been tried in an experimental way, and has as yet attracted very little attention. This machine belongs to the class of continuous plane washers, and is a developed form of the Brunton traveling belt (devised in England forty years ago), to which a lateral shaking motion has been imparted. It consists essentially of an endless rubber-coated canvas belt, which is set on rollers at a slight inclination, so that the washing surface can move upward against a flow of slime and wash water. A sidewise oscillating or vanning motion is given to the belt to promote a settling of the mineral, and to distribute the sheet of slime evenly over the washing surface. The deposit on the belt is carried over a revolving drum at the head of the machine, and forms the "concentrate;"

the slimes flowing over the lower drum are the tailings; the washer produces, as a rule, no middlings.[1]

Very fine head mineral will naturally settle at a lower point on the belt than coarser mineral of the same kind, compared with which it will require more time in reaching the head drum of the machine, and will be subjected for a longer period to the flow of slime and wash water, with all the greater chance of being carried over the lower drum by the current. On this account the vanner does not seem as favorable for the production of low grade tailings as the outward-flow round table on which the force of all the currents diminishes as these spread over an increasing area toward the lower part of the apparatus. The characteristic vanning motion of the machine, however, compensates more or less for its defect. It facilitates the flow of the slime, particularly that of the larger, easily rolling particles of gangue, while the fine mineral, when once deposited on the belt, adheres closely to it, and is but slightly affected by the lateral shaking motion.[2] The favorable effect of vanning upon the flow of slime permits the use of very concentrated slimes and little wash water, and insures a capacity which is large compared with that obtainable on a plain traveling belt. One machine commonly treats 6 tons of fine stamp-meal per day (not including the re-washing of tailings if such be necessary), though a capacity of 12 tons has been attained under highly favoring conditions. Two machines generally receive the slimes from a battery of five stamps.

A form of continuous end-percussion machine was patented

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[1] That the vanner is not well adapted to form a "middling" product would in foreign mills frequently prove a serious objection to it. In western practice, however, the finely divided gold or silver bearing mineral is usually the only concentrate sought, and middlings of other minerals have but small value. The finely stamped slimes of meal and pulp are there led, without grading, from the batteries directly to the slime washers. By passing the material successively over two vanners, it is possible, under very favorable circumstances, to extract 90% of the precious metals contained in the ore. The saving of 80–85% is a more usual result in practice. It must be borne in mind that as a rule the ore-slimes in the West are of a very much higher grade than those in Europe, and it is found from a number of authenticated cases that the western tailings which carry only 10–15% of the "mineral" contained in the slimes are as rich and even richer than many of the unwashed slimes that are treated in foreign mills. The home practice could without doubt be improved in many cases by introducing a preparatory hydraulic classification.

[2] This mode of action will suggest at once why the vanner has found particular favor in the treatment of fine stamp meal and pulp—the *mineral* being nearly always reduced by stamping to a finer condition than the *gangue*.

in France in 1865,[1] and may be described substantially as a Brunton belt traveling over a suspended frame to which an endwise percussive motion is imparted by revolving cams at the head of the machine. The machine has not been seen in operation by the writer, but it is evident that the effect of the endwise percussions on the slime flow must be the same in kind as that produced on intermittent end-percussion tables. There must be a succession of momentary arrests or retardations in the flow of slime over the belt, and these frequent interruptions prevent the attainment of an excessive maximum velocity of flow which otherwise would be reached on any surface, having more than a very gentle dip, soon after the slime had started on its course down the incline. The loss of fine mineral is, therefore, minimized by the use of the end percussion, and the independence realized, to a limited extent, between the dip of the belt and the velocity of the slime flow, permits of setting the belt at a comparatively steep pitch, promoting the rolling of the coarse gangue grains and increasing the capacity of the machine. When run at high speed, the end percussions additionally liven up a very concentrated slime, and exerting in this respect, like the lateral vanning of the Frue machine, a greater effect upon the coarse gangue grains than upon the finest mineral particles, the end-shaking motion promotes the separation of these products near the tail end of the concentrator.

Among the class of plane surface washers this continuous end-percussion machine certainly possesses excellent features, and its failure to secure a prominent position in slime dressing practice cannot be ascribed to defects in principle.[2]

Tossing Keeves.—The English dolly-tub, or tossing keeve, which in its earliest form was a simple hand tub, is still used in many English works in treating imperfectly sorted slimes. In its present form the apparatus usually consists of a fixed tub, within which a stirrer rotates about a vertical axis. Out-

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[1] By Parent, Schaken, Caillet & Co.

[2] In American practice, the Embrey concentrator is a continuous working slime washer, designed on very similar principle to that of the French machine. It consists of an endless traveling belt, to which an endwise shaking motion is imparted by eccentrics at the head of the machine. The effect of the shaking motion upon the slime is the same in kind as that produced by the gentle percussive motion of the foreign machine, so that the concentrator, if well designed as regards length of washing surface, speed of travel, etc., should give, comparatively speaking, very favorable results.

side of the tub is a small mechanical hammer which, when in operation, taps the tub fifty to eighty times a minute. The tub is one-third filled with water, to which the fine ore or concentrate is gradually added until the consistency of a thick slime is obtained. The stirrer is kept revolving during the mixing period—about 15 minutes—and imparts to the slime a rotary motion. Then the stirrer is raised out of the keeve by a light tackle, being at the same time disengaged from the transmission gearing, and the hammer is set in operation to promote the settling, which requires 15–20 minutes. The gradually diminishing rotary motion of the water has the same effect on the slime particles as a deep current of decreasing velocity, and the material settles in layers of equal-falling grains. After the deposit has settled, the top water is tapped off, and the material is shoveled out of the keeve, forming several well-defined products—a rich *bottom*, a class of middlings, or *krazer*, and a fine, barren *top*.

Sand slime, unless of the finest grade (1.4–1 mm., or $\frac{1}{16}$–$\frac{1}{25}$ inch), is too coarse to be advantageously treated by "tossing," but meals are well suited for it; pulp slime, on the other hand, is too fine, requiring too much time for depositing.

The headings from Cornish buddles are always purified by tossing, if a sample lot is found, upon vanning, to contain very fine dirt mixed with rich mineral of medium fineness. The washing of enriched tin concentrates at the Dolcoath mill, at Cambrae, Cornwall, may serve as an illustration of the practice. These concentrates are the headings from a set of round buddles; they carry 60–65 per cent. of *tinstone*, mixed with a certain amount of rock gangue and some very fine red oxide of iron, produced by the roasting of the pyritiferous tin ore in meal size. The material is subjected to "rag-tossing" and yields bottom, krazer and top products, each of which is rebuddled—the bottom product but once, the others several times. By buddling the *bottom*, a separation into heads, middlings and tailings is obtained; the two poorer (and coarser) products are re-treated, but the headings are subjected to "clean-tossing," which is repeated four or five times, producing finally a marketable concentrate, carrying over 90 per cent. of cassiterite, and a poorer *top*, which is re-tossed or re-buddled, according to its fineness. The purpose of the treatment is to separate by repeated buddling the coarse gangue particles from the finer mineral, and, by repeated tossing, to

purify the concentrate from the impalpable ferric oxide which holds to the much heavier *tin* with surprising fixity.

The purification by tossing is particularly applicable to meal concentrates, which, owing to imperfect grading, contain a considerable percentage of very fine, usually slaty dirt, that adheres closely to any washing surface, and cannot be separated from the mineral on tables without incurring considerable loss. The tapping of the hammer on the side of the keeve is very advantageous; by shortening the time required for settling, it extends the application of tossing to much finer sizes of material than could otherwise be profitably treated.

Comparisons between Different Slime Washers.—In discussing the relative merits of the four prevalent types of slime washers—the fine jig, the round table, and the end and side percussion tables—the results obtained with these apparatus in various mills have to be most carefully examined lest they prove misleading. Even experiments systematically conducted in a single mill can easily lead to error if they be made the basis for broader generalizations than a rigid adherence to the conditions of the experiments permits. In dressing any given kind of slime, the capacity of the slime washer, the degree of enrichment of the concentrate and the quality of the tailings are interdependent qualities, all three of which must be known in forming an opinion as to the efficiency of the apparatus. When in this connection, it is considered that the external influences which affect the slime treatment are in no two mills identical, and when the great diversity in the character of ores and the modifications of working arising from this source are taken into account, the necessity for a very circumspect consideration of the relative merits of any particular treatment must be evident. Dressing establishments which are situated near smelting works generally run for clean tailings rather than for high grade concentrates, and this is especially the case when the smelters treat several kinds of concentrates with gangues that are mutually self-fluxing. It is not unusual to find in the dressing of an ordinary galena-blende ore that the slime middlings of blende have to be concentrated by repeated washings to a higher percentage of metal than the galena headings, mainly because transportation to the zinc smelters is over a greater distance and more costly.

Meal jigs have been used at Clausthal for treating two classes of graded slime between the sizes of 1 and 0.5 mm. For the

purpose of experiment, slime of the same class was treated on central discharge tables, 5.5 m. in diameter. The advantages in favor of the table proved to be smaller cost of the apparatus, fewer repairs, less motive power, less attention required from workmen, greater capacity and a better separation. More galena was obtained in the headings and less in the blende middlings; the tailings were immediately re-treated on a lower table. The better separation is partly due to the fact that the ore gangue consists of quartz and spathic iron; much of the latter settled upon the mineral bed of the jig, forming after a time a compact layer which impeded the sorting and necessitated frequent stoppages. After obtaining such results, some of the jigs in the slime department were replaced by central discharge tables, and all will eventually be superseded by them.

On the other hand it is reported[1] that at Scharley, in Upper Silesia, in washing a blende ore carrying some galena, 34 round buddles have been supplanted by six continuous sand and meal jigs, and that thereby the amount of lead in the zinc concentrate has been reduced from 1.5 per cent. to 0.1–0.2 per cent. The gangue in this ore is dolomitic, and the slime is therefore "tough;" sand and coarse meal slimes of this ore are thoroughly agitated by jigging, and hence more easily cleansed from the fine adherent gangue than by the quieter action on buddles or tables. Such and similar special cases, found only by actual experience, will warrant the use of jigs for the sizing of graded meal slimes, but as a general rule the jigging of material finer than 1 mm. has not been satisfactory. The Clausthal works were regarded as leading advocates of very fine jigging, and their change to the double inward and outward-flow tables will probably check further movement in that direction.

Side-percussion, or Rittinger, tables have been used with signal success at some mills, while they have been discarded as worthless at others, notably at some of the large government mills at Przibram. The constant attention required at the machine, unless the slimes are of a very uniform character, is a point always urged against it. The principal reason why the apparatus has frequently met with poor success lies probably in its application to the treatment of slimes for which it is unsuited. All the mineral upon the the table must be rolled over to one

[1] Judges' Reports to Centennial Commission, 1877. Vol. I., p. 293.

side by lateral shocks, and to move in this manner with facility the particles must not be too small. Very fine pulp, as is well known, is peculiarly adherent to any washing surface, and therefore not suited for treatment by this method.[1] Coarse material, on the other hand, is apt to roll too easily under the effects of the shocks, and so increases the difficulty of producing a good separation. Experiments with the Clausthal ore (described on page 17), upon a table having a fine grained wooden surface, showed that slime containing mineral coarser than 0.5 mm. and finer than 0.25 mm. ($\frac{1}{50}-\frac{1}{100}$ inch) could not be satisfactorily treated. Washing surfaces which are made of other material alter these figures to a certain extent; but the practice of treating pulp slime on side-percussion tables, even though these may be fitted with very smooth marble surfaces, cannot be generally recommended. The Rittinger tables are best adapted for washing graded coarse meal slimes (which carry *mineral* within the limiting sizes above named), and their use is profitably extended to finer material only when very highly concentrated headings are required. They have been very successfully used in extracting fine galena from blende middlings of meal, and fine sand, jigs. The side-percussion table has sometimes been styled the connecting link between the jig and the round table. It does occupy an intermediate position in some cases, though the combination of inward and outward-flow round tables has been found suited to meals as coarse as any treated on the percussion table. The special adaptability of the latter to collect headings which are contained in very small quantity in the slime, and the high degree of concentration which it can effect (often at the cost of producing a large class of middlings), are elements fully as important as size in determining which type of apparatus to employ.[2]

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1 The adherence may be partly due to a feebler current very close to the table surface, but its main cause lies in the small distance between the centre of gravity of each particle and its supporting base, and consequently in the short leverage of any overturning force.

2 Compared with the Frue vanner, or the Embrey concentrator, the side-percussion table seems better adapted for treating coarse meal slime, and less so for pulp slime. Both classes of machines produce high grade concentrates. The percussion table has the advantage of producing middlings, and is cheaper, but the endless belt machines consume less water and less power. In treating *fine meal* slimes the concentrations of both kinds of machines and their working capacity and saving of head mineral should be about the same, so that preference for washing this class of material will be given to one or the other machine according to the relative importance of the above named characteristics.

The end-percussion table still maintains a prominent position, notwithstanding the manifest disadvantages of working without a continuous discharge of each of its products. It is a very simple machine to manage, and both the headings and middlings can by successive operations be easily enriched just to the grade required. For this reason the table is often used in working up blende products. The blende, to obtain a favorable market price, must not carry over a certain percentage of galena, and this condition is readily attained by successive operations on the table. Experiments on Przibram quartzose ore (previously described) showed that the table was not suited for coarsely stamped *sands*, 1–1½ mm. in diameter. For such material a sand jig (with 6–8 mm. stroke, and stayed water) effected a much larger saving of head mineral. Similar results have been obtained elsewhere: in the blende department of the Ems mills end-bump tables have been replaced by four-sieve jigs for washing two graded sand classes between the sizes of 1½ and 1 mm. Further experiments[1] for the purpose of comparing the results obtained in washing very fine pulp on a sweeping table, a Rittinger table (2.5 m. long) and an end-percussion table (4 m. long and worked with strong shock) proved the superiority of the last-named machine in point of capacity, mineral saving, value of products, consumption of water and cost. The results would have been far more interesting if a round table had been drawn into the experiments, but such as they are they will serve as confirmatory evidence that the Rittinger table is not adapted to dressing fine pulp.[2] The experience of several mills is that tough (slaty or calcareous) slimes cannot be successfully treated on the end-percussion table. For washing quartzose meal, however, and pulp slimes, especially in blende dressing, this machine is likely to remain in use for some time in the mills where it is now employed. Its gradual abandonment, however, in favor of continuous machines may be inferred from the fact that it has not been adopted in any of the large first-class establishments which have been erected in recent years. The prospect of its permanent adoption where the value of labor is high can never be entertained.

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[1] Oestr. Zeitschrift f. Berg und Huettenwesen, Vol. XXVII., p. 101.

[2] The Schemnitz practice, as indicated under the subject of *Hydraulic Classification*, shows that the end-bump table is there regarded as filling an intermediate position between side-percussion and round tables.

Round tables, which are the machines most commonly used for washing meal and pulp slimes, possess a wide range of utility. The inward-flow table treats coarse meal satisfactorily, while the outward-flow or cone table has, under many circumstances, proved to be the most suitable continuous apparatus hitherto employed in foreign practice for washing fine pulp. A round table generally produces, in the first operation, marketable concentrates, though not of as high a grade as the similar product from a side-percussion table; middlings which, if not too fine, can be treated to advantage on a Rittinger machine, and tailings that are delivered at once to a lower round table. Decided advantages for this machine are low cost and small items for repairs, labor and power. Fine pulp, carrying less than 2½ per cent. of very low grade argentiferous galena has been washed on it with profit.[1] It seems beyond doubt that on the Continent the round table will remain the leading type of slime washer until some new apparatus of radically different and better design shall supplant it.

Crushing and Drying of Concentrates.

Many foreign smelt-works require that concentrates be reduced to a 3 mm. (⅛ inch) size before subjecting them to metallurgical treatment. Accordingly dressed concentrates from jigging are sometimes drained on screens,[2] or they are dried by simple exposure, and then passed, together with "mineral" from hand sorting, to a set of reducing rolls, or to a Chilian mill.

Concentrates from the slime department naturally require no further reduction, but they must be dried in the dressing mill in order to reduce the cost and diminish the losses of transportation. A very simple contrivance for separating most of the water from these fine products is a tub which is tapped on the outside by a self-acting hammer, or a box balanced upon a horizontal axis and rocked back and forth by an eccentric, so that it taps the floor at each stroke. The deposit settles down and compacts well in either kind of apparatus; a layer of water several inches deep forms on the surface and is drawn off. The dried mineral contains from 1½–4 per cent., and in

[1] Such working results in treating pulp have not been reached, to the writer's knowledge, with any other slime washer.

[2] Endless belt screens with an up-and-down shaking motion have been introduced for draining coal after jigging, and are proposed for the same purpose in large ore dressing works.

some districts, even 10 per cent. of water. It is shovelled out of the drying-box, is sampled, assayed and weighed, and is then packed in bags or loaded for shipment.

Losses in Wet Dressing.

By observing suitable methods for minimizing the production of dust in hand-sorting and crushing, and for saving whatever mineral is comminuted in those operations, the losses in dressing ore classes of nut, pea, and coarse sand sizes have, in present practice, been reduced to a minimum for all ores in which the difference between the specific gravities of the constituent minerals is sufficient to insure the success of ordinary dressing methods. In the treatment of coarse ore, the only mineral or metalliferous portion, so-called, that actually goes to the waste dump is such as is locked in "barren" products from hand-sorting and jigging—material which, by simple calculation, is found too poor to be profitably subjected to the further processes of slime treatment.

In slime dressing the mineral losses have been reduced by substituting new comminuting mills for stamps, and by introducing continuous processes of concentration in which the production of dry float mineral is avoided. The middlings and tail products from the slime tables are re-washed as often as they will pay for the treatment, but nevertheless the losses continue to be very high. At Clausthal, where work is particularly directed toward obtaining clean tailings, rather than high grade concentrates, the average contents of the ore subjected to stamping and washing are 3–4 per cent. of argentiferous lead; the waste tailings are not regularly assayed, but their mineral contents are estimated by the difference between the percentage of mineral in the ore and the quantity obtained in the concentrates, and such calculation has shown repeatedly that the loss amounts to fully 50 per cent., and is frequently nearer 60 per cent., of the mineral carried in the crude slime. Results obtained in a number of leading mills show that the dead slimes run 2–4 per cent. of lead, and that the losses range from 45 to over 60 per cent. of the mineral contained in the crude slimes. Such figures raise the proportion of total losses incurred in the whole course of dressing to 12–16 per cent. of all the mineral in the ore.[1]

[1] The best American practice can point to a saving of 90 per cent., or sometimes of even a greater proportion, of the mineral carried in the ore, even when it is in a

From time to time brief statements appear in the technical journals to the effect that some mill has increased the capacity of its tailing basins, or has begun to wash fine pulp that had formerly been discarded. So, for example, it is noted that additional large round tables have been put into operation at Clausthal, and these collect from the finest settlings of a very extensive labyrinth forty-six pounds of concentrate, running 50 per cent. of lead, per cubic meter of pulp. The cost of the washing is found not to exceed 12½ per cent. of the value of the concentrate obtained from what heretofore has been a waste product. Fifty per cent. of the mineral of that product is now saved by the new, extended washing facilities, and the final tailings to-day carry a little less than 2 per cent. of lead—a result which is as good as any that has come to the writer's knowledge.

Such journalistic notices indicate that the managers of dressing works are alive to treating all material that can possibly yield a profitable return with existing methods and with the means at their command. Losses, then, can rarely be ascribed to neglect in practical working; their source is to be traced, rather, to defects inherent in the prevailing systems of concentration.

One of the most thorough investigators in the field of ore dressing has been Sparre, a Prussian mining official, who devoted much of his time to slime washing, which is the most difficult branch of the subject. He experimented with an inclined glass table, of small, model size, inclosed in a glass tube which formed one arm of a V-shaped pipe, with flexible, rubber joint. With this apparatus he was enabled to regulate the velocity of a flow of water or of slime over the washing surface independently of any inclination of the latter.[1] He used the apparatus to confirm his theoretical investigations, and he demonstrated that when ore particles of the same specific gravity are exposed on an inclined plane to a shallow stream of water, a certain dip[2] of the washing surface will be found

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state of very fine dissemination, but in making comparisons with foreign practice, the very low grade of most of the foreign ores must be borne in mind, and only the absolute (not the relative) losses should be considered.

[1] *Sparre*, Zur Kritik der Separation, p. 26, illustrated description.

Althans, Judges' Reports on the Centennial Exhibition, Vol. I., p. 228, illustrated description.

[2] This dip is not to be confounded with the angle of repose in cases of sliding friction.

which allows the particles of one particular diameter to offer the greatest resistance to the current—or, differently expressed, with a gradually increasing velocity of current, but with a constant dip of the table, all the particles which are either larger or smaller than those of one special size will be carried off the table, while the particles of this one-size class will remain deposited, and can only be removed by a still further increase of the current after all the other material has been washed away. The angle or dip of maximum resistance increases as the size of the particles diminishes. For a given ore-size, this angle is independent of the specific gravity of the particles; but the minimum current velocities which, for any given dip of the table, suffice to carry two equal-sized bodies of different specific gravities off the washing surface are proportional to the square roots of those specific gravities. In sizing a class of equal-falling ore particles on the experimental washing table of glass, the larger (specifically lighter) particles could be separated from the smaller headings with a minimum quantity of water by setting the dip of the washing surface at the angle of greatest resistance of the smaller size class.

These important principles, demonstrated by experiment, could not be directly incorporated in a washing machine designed for practical use, because the washing surface of such an apparatus would have to be covered over, and would, therefore, by precluding all observation, prevent adequate control of the separation. Sparre, in his further studies, concluded that the most practical way for establishing an independence between the dip of the washing surface and the velocity of the water current on a continuous machine, lay in the use of an open or uncovered table, to which a continuous or intermittent rotary motion should be imparted.[1] This motion, communicated to the slime, could be accurately regulated on a table of any dip so as to develop in each ore grain a certain centrifugal force, which, combining with the force of gravity acting in the direction of the dip, would produce a definite resultant for each different class of particles, and so permit of a good separation. Through practice it was soon made evident that the dip of the table should not be great, but rather so calculated that a moderate centrifugal force would suffice to produce the

[1] A continuous motion proved best for washing hydraulically graded material, while an intermittent motion was preferred for the treatment of fine, ungraded slime.

necessary resultant. An intermittent oscillating table,[1] designed on Sparre's principles, is reported to have produced excellent results, but the machine has never obtained prominence.

None of the prevalent types of washing tables fulfils the conditions shown by Sparre's theories and experiments to be essential to the best attainable results in slime washing. Everywhere the dip of the tables is used to regulate the velocity of the current which flows over them, and it cannot, therefore, serve to control the rolling motion of the ore particles.[2] The angle of inclination is smaller than what Sparre found most suitable when the velocity of the current was independent of the dip, and since on ordinary tables and vanners this velocity must diminish with the fineness of the slimes that are treated, the disproportion in respect to inclination correspondingly increases. In common practice, the slime particles which are charged upon a smooth washing surface pass over the greater portion of it with but an imperfectly governed sliding movement, instead of advancing with a definite, well controlled rolling motion, and a course of many feet is required to complete a separation which a very much shorter space should easily accomplish. Blanket cloths and designedly roughened surfaces are frequently employed to prevent the sliding of the mineral particles. Their use for this purpose is attended with a certain degree of success, but they interfere in other ways with the efficiency of most separations.

Slime dressing, in the present state of its development, is clearly wanting in machines that incorporate all the principles upon which the art is based; there are at present no washing tables—or, at least, none such generally known or used—yielding results which can be regarded with complete satisfaction. Radical improvements must still be sought for, and these, developed in accordance with the principles above summarized, should lead to the design of smaller, less expensive and more efficient machines.

More than thirty years ago, Pernolet noted that the laws governing the free fall of bodies in water do not hold good for very fine particles, on account of the increasing influences of the adhesion and viscosity of the water compared with the

[1] Constructed by the Humboldt Machine Shops (then Sievers & Co.) at Kalk, near Cologne.

[2] Continuous working machines of the end-percussion, or end-shaking, type secure, but only to a very limited degree, the desired independence between the dip of the washing surface and the velocity of the slime current.

weight of the small mineral body. This observation seems to have remained for a long time without fruitful issue in the treatment of fine slimes. Pulp is still washed on round tables in very much the same manner as coarse meals. The palpable result of so defective a system is found in apparatus of extravagantly large size, with small capacity and low efficiency. In view of these facts, the development of dry ore dressing,[1] at comparatively recent date, is particularly interesting. On general principles it is evident that a separating medium of high density is favorable in dressing operations when the influences of its viscosity and its adhesion to the ore particles are so small that they may be neglected.[2] When, however, adhesion and viscosity can seriously interfere with the movement of very small ore particles acted upon by gravity, the denser medium may with advantage be replaced by one that is lighter but less adhesive. In such cases "dry," or pneumatic concentration profits by the very slight adhesion of air to the ore, and in carrying out the separation on an air jig, there is no troublesome return current moving downward through the ore-bed, and practically no inertia of the separating medium to limit the speed of the machine. A speed of 500 strokes per minute has been attained and good separations effected.

Pernolet found, in experimenting with bodies falling through water confined in tubes, that the resistances due to the walls of the tube were perceptible when the moving body occupied only $\frac{1}{80}$ part of the section of the tube.[3] With larger grains, or with narrower tubes, the influence of the tube walls becomes more appreciable, and the laws of movement of the bodies are no longer those of unobstructed fall through water. At a much later date Bartlett pointed out, in a very interesting paper,[4] how some of the remarkable results obtained by Krom in experimenting on pneumatic separations were effects of that same cause—namely, resistances due to the enclosing walls of

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[1] By Krom, and Paddock, and others in this country.

[2] If the ratio of the specific gravities of two equal sized bodies be as 3 : 7, the ratio of their falling weights in water will be as (3—1) : (7—1) or as 2 : 6, showing a gain in the ratio in favor of hydraulic, over air, separation. In the case of saline water, such denser medium would be still more favorable for the separation, aside from the fact that salt water is more liquid (or less viscous) than fresh water.

[3] Pernolet; *Études sur la Préparation Mécanique*: Annales des Mines, 1851, tome XX., p. 564.

[4] Bartlett: *Action of Small Spheres of Solids in Ascending Currents of Fluids, and in Fluids at Rest*: Trans. Am. Inst. of Mining Eng., 1877–78, Vol. VI., p. 415; Engineering and Mining Journal, 1877, Vol. XXIV., pp. 102 and 129.

the air current; or, in other terms, he explained how a rising current of air in a narrow tube could easily separate two different mineral grains that were equal-falling in water, by the buoyant action resulting from a difference of gaseous tension in the tube above and below the larger, specifically lighter, grain which the air current sustained. The influence of the enclosing walls is preserved in Krom's pneumatic jig, but the delicate means of separation thus obtained necessitate correspondingly great uniformity of the blast and very accurate sizing of the ore. Slight variations in either of these quantities will change the whole effect of the dry treatment, and this is one of the difficulties which has to be met and overcome in practice. Coarse ore cannot be dressed without air currents of very high velocity, and, moreover, since the important relative advantage of air over water—the slight adhesion of the former—disappears in treating all but fine material, it seems almost certain that coarse pneumatic jigging will be restricted to dry regions where water is expensive.

Dry jigging has recently attracted some attention for the treatment of the finely mineralized tin ores of Cornwall. A promising field for it, though one in which several leaching processes have of late been successfully advancing, is presented in the dressing of gray copper and of fine, brittle sulphide and telluride ores carrying the precious metals—ores that cannot be treated without great loss on washing machines.

As to the extent to which pneumatic concentration may in the future replace slime washing or modify the treatment in a "wet" mill, it would be premature, at present, to hazard an opinion. When the large mineral losses incurred with present methods are taken into account, and when it is also considered that even the most perfect slime washer, designed in accordance with the laws of the free movement of bodies in water and on inclined surfaces, could never treat pulp slime to complete satisfaction (for the very reason, as Pernolet has shown, that the pulp particles are not governed by those laws), then the necessity for some radical change appears to be very evident. Such change and improvement may in many cases be found in the adoption of pneumatic concentration, though certainly not before the new system has been further developed by careful study and experiment.[1]

[1] The practice now pursued in the West, of mixing with water the dust collected from dry concentration, and washing the resulting slime on vanners, is di-

SPECIAL DRESSING TREATMENTS.

Reference has at several points been made to special dressing operations based upon some particular property in the ore, other than the specific gravity of its constituents, and it now remains to give a short account of such processes, prefacing here that their introduction is still quite limited, but that the growing spirit of independence in experiment and practice promises further developments and a wider field for their application.

Calcining.—Very tough rock mined in large pieces is sometimes piled together with fire-wood in open heaps; light is set to the fuel, and when the ore is thoroughly heated, the fire is quenched with water. The rock is fissured or rendered "tender" by this treatment, and then is crushed with comparative ease. The practice of calcining, a very old one, prevails in the tin districts of the Saxon Erzgebirge, where finely disseminated tinstone is carried in a tough granitoid gangue.

A recent application of calcining is reported from Oberlahnstein, on the Rhine. A dressing product of blende and spathic iron is heated to redness, and then dropped into water; the spathic iron is thereby thoroughly disintegrated, and falls into very small particles; an easy separation by sifting then follows. This method promises to be a very useful and valuable one, though probably not applicable to ores of a very finely intermixed texture.[1]

Roasting.—An oxidation by means of roasting is sometimes resorted to after two or more mineral species of nearly the same specific gravity have been concentrated out of a common gangue, and are to be separated from one another. In Cornwall, for example, the concentrates of tin ore carry, after repeated buddling, 45–55 per cent. of tin, the balance being principally iron, sulphur and arsenic, in the forms of arsenical pyrites and spathic iron. The specific gravities of these minerals are—cassiterite, 6.4–7.1; arsenopyrite, 6.3; siderite, 3.9. By roasting, all the minerals but the tin are decomposed, and the roasted product consists practically of tinstone and very

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rectly opposed to the essential condition to which pneumatic jigging should, it seems, look for success, namely, to the efficient dressing of extremely fine material.

[1] For further examples of calcination as applied to ore dressing, see THE SCHOOL OF MINES QUARTERLY, Vol. III., p. 55.

fine sesquioxide of iron. The specific gravity of the latter is 4.5–5.3, so that its separation from the tin by further buddling is quite practicable. A small proportion of stannic oxide, not over one-half per cent., is sulphated in the operation, and lost in the subsequent washing. At some of the mills the arsenic fume is saved as a commercial product. The oxidation is performed in a Brunton mechanical roaster, which has a circular, slightly conical, rotating hearth, set on a vertical axis in a round reverberatory chamber. The concentrates, after being well dried upon the roof of the furnace, are charged into a hopper from which there is a continuous feed on to the hearth; the latter moves with a speed of six to eight rotations per hour, according to the quality of the charge and the length of time that the same is to remain in the furnace. The material falls upon the centre of the hearth, and is heated to a dull red by two reverberatory fires. It is gradually moved outward, to the periphery of the hearth, by a set of fixed, horizontal arms, set radially and furnished with vertical flukes which stir the slowly revolving bed of ore. The discharge, by means of a scraper, is also continuous; the roasted ore falls into a closed receiver and is wetted down (to avoid arsenical dust) before further treatment. The ore requires six hours in passing through the furnace. The roasted concentrate is of light gray color, but shows the characteristic red of ferric oxide as soon as it is suspended in water. For a similar treatment of tin ore in Saxony, long reverberatory furnaces are used, but they certainly cannot be worked as economically as a mechanical roaster, of which the Cornish furnace is one type.[1]

Another case of roasting, and one of special interest because it bears directly upon the difficult problem of treating concentrates carrying zinc blende and iron pyrites, is found in the ore dressing practice at Iserlohn, in Westphalia. Blende and pyrites occur in many ores associated with other minerals; their specific gravities are respectively 4 and 4.9, and all efforts to effect their complete separation by mechanical washing have been practically unsuccessful. In ordinary dressing, a large

[1] In some few cases roasting is combined with further chemical reactions: to remove the tungstate of manganese, wolframite (sp. gr. 7–7.5), from the tin concentrate with which it is not unfrequently associated, soda is mixed with the ore, and in roasting a double decomposition takes place; sodium tungstate is formed and leached out with water. Salt also has been used with tin concentrates, to chloridize impurities of copper and furnish an alkaline base for tungstic oxide. *Vide* Kerl, *Allgemeine Huettenkunde*, pp. 496–7.

proportion of these two minerals is obtained as one product, both from jigs and from slime washers, and though the concentrate may run high in zinc as well as in iron, yet the mixed product sells to no profit. At some dressing mills, even among the foremost, this product is discarded, or perhaps accumulated on the chances of future realization. At Iserlohn the product in question is shipped a few miles by rail to zinc smelting works, and is there roasted. Almost all the blende is thereby converted into zinc oxide, and the pyrites into ferric oxide. The roasted material is jigged with a little extra care, and a fair separation obtained. The separated products are readily marketed, and the mode of treatment is said to be profitable. In working the process, it has been found that the roasting should not be carried beyond a certain limit; 3 per cent. of sulphur must remain in the material after the roast, in order to preserve a certain coherence in the zinc oxide. But notwithstanding this precaution, a very appreciable amount of zinc is lost in washing the oxidized minerals, so that this method cannot be regarded as altogether satisfactory. The process was suggested by the finely crystalline, as well as closely banded, structure of the blende and pyrites. To disintegrate the ore by any ordinary mode of comminution would have necessitated a very fine reduction with the attendant losses. By the action of heat, however, the ore is broken along the planes of banding, and the roasted charge is obtained largely in the form of small tabular pieces.

Separation by Friability.—A practical method for effecting the separation of some minerals by means of their different degrees of friability has been devised by Fr. Buttgenbach, at the Lintorf lead mines in Rhenish Prussia.[1] The process is there applied to the separation of blende and pyrites. The worthless jig concentrates of these minerals are passed through a Vapart centrifugal disintegrator[2] in which the ore particles, impelled by centrifugal force, are hurled against the circular walls of the machine and then drop through a discharging chute into an ordinary sizing trommel. The speed of the apparatus—about 350 revolutions per minute—is regulated by experiment so as to develop a velocity in the movement of the particles sufficient, upon impact, to break up the blende, but not the

[1] *Vide*, the SCHOOL OF MINES QUARTERLY, Vol. III., p. 55.

[2] The Vapart Centrifugal Disintegrator, invented at Vielle Montagne: Illustrated description, *Revue Universelle*, 1881, Tome X., p. 552.

pyrites. The fine broken blende is separated by sizing from the coarser pyrites, and an intermediate product—blende and pyrites of medium fineness—is generally formed and returned to the disintegrator. This machine breaks up 2½–3 tons of stuff per hour. The blende sand carries 50 per cent. of zinc, and is a valuable product, which is also true of the pyrites. At Lintorf it is found that for satisfactory working, the material must be coarser than 2.5–3 mm. (⅛ inch) and the best results are obtained with 5–6 mm. ore. In developing the process, it will doubtless be found advantageous to run for a comparatively large intermediate product which will then be passed through a second machine revolving at a higher velocity than the first. There are many ores to which this method might probably be well applied, and further accounts concerning it will certainly be awaited with interest.[1]

Electro-Magnetic Separation.—By careful roasting at a dull red heat (600°), pyrites can be converted into magnetic oxide of iron, which is readily extracted with the magnet from a silicious or calcareous gangue. About twenty-five years ago attention was directed to this means of concentrating pyritiferous ores, and a practice of magnetic separation was developed for very low grade ores of copper and iron pyrites in Northern Italy. An allied practice is still found in operation in the Lill dressing works at Przibram, Bohemia, where spathic iron is separated from zinc blende by gentle roasting (which converts ferric carbonate into a magnetic oxide), and subsequent extraction of the iron with magnets.

The early magnetic separator consisted of a row of fixed horseshoe magnets, beneath whose poles was passed a layer of ore thinly spread upon a traveling belt; or, more frequently, the magnets themselves were movable, being set radially in a horizontal, rotating drum, with their poles projecting from its circumference.[2] In the course of one rotation of the drum either the poles were passed through the pulverized and, if need be, roasted ore, or the latter was fed slowly down upon

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[1] An example, in Western practice, of a concentration based upon friability, is found in the method formerly pursued at Lake City, Colorado. The ore after being crushed between soft steel rolls was sifted, whereby the friable, finely comminuted, rich mineral was directly separated from the tougher and coarser quartz. *Vide*, the SCHOOL OF MINES QUARTERLY, Vol. IV., p. 3.

[2] Illustrated Description of the Lill Magnetic Separator: Transactions Am. Inst of Mining Engineers, 1881, Vol. IX,, p. 420.

the cylindrical revolving surface. In both cases the attracted particles, being held by the magnets, were raised to a point where they could be conveniently brushed away from the poles and allowed to fall into special receivers.

The objection urged against one and all of these machines was their small capacity—they could treat only a few hundred weight of ore per day—and this drawback prevented their general adoption. It was not until large electro-magnetic separators were constructed by French and German inventors that experienced ore dressers found it practicable to introduce magnetic concentration in their works. Of late years several excellent magnetic plants have been erected.

One form of separator devised by Dr. Werner Siemens, for a mining company working an ore of oxidized zinc and spathic iron in Spain,[1] has the form of a rotating trommel, or drum, set on a slightly inclined axis. The drum consists of a number of annular iron discs, separated from one another by rings of brass. The discs are held firmly in position by iron strips, which bind them together on their outer peripheries, and so serve to convert them into a series of horseshoe magnets whose poles form the inner surface of the drum. The end discs of the drum are secured to iron frames or spiders, which support the whole on an insulated steel axle. The magnetization of the apparatus is effected by passing a current through a circuit of wire, which is wound around the brass rings, between the annular discs. Pulverized ore is charged, after roasting if necessary, into the inclined drum, and slides gradually toward the discharge end. The magnetic particles adhere to the sides of the drum and are raised above the axle to a line where a fixed brass scraper detaches them. They fall into a small brass leader set around the axle, and are discharged by a brass screw conveyor. A refinement is introduced into the apparatus by surrounding the rings at the head of the drum with fewer coils of wire than those at the discharge end. This end, therefore, becomes the most strongly magnetized, and picks out of the ore such feebly magnetic particles as pass through the head of the drum without being removed. Such arrangement distributes the work of separation between all the rings of the drum and notably increases the capacity of the apparatus. Small machines, 3½ feet long,

[1] Illustrated description: Werner Siemens, *Gesammelte Abhandlungen*, 1881, pp. 537, *et seq.* (Julius Springer, Berlin).

2½ feet in diameter, are claimed to treat 1–2 tons of ore per hour, according to the ease or difficulty of the separation.

An interesting magnetic plant is in operation at the Friederichssegen Mine near Oberlahnstein, Nassau, treating roasted jig products of blende and spathic iron. Ore up to 5 mm. (⅕ inch, or nearly 3½ mesh) goes through the process, but the size best suited for it is found to be that of 3 mm. (⅛ inch, or 6 mesh). Roasting is performed in a long Freiberg roaster and in two short reverberatory furnaces. The latter give the most satisfaction; two men running the two smaller roasters for twenty-four hours can put through as much ore as six men at the long one, for in the small furnaces there is no periodical movement of the charge along the hearth, while the efficiency of the roasting very nearly equals that reached in the large one. The roasting in the small furnaces lasts two hours, during which time the escape of carbon dioxide from the ore is noticed by the sound of slight crepitation. At end of the operation the charge is at a dull red heat, and is tested with a hand magnet. The roasted ore is elevated to bins, which supply the magnetic separators. There are four of these machines—two on one floor of the mill and two on the floor below. The upper machines receive equal quantities of ore, and both produce an enriched zinc concentrate and an enriched iron concentrate. Each of these products is fed through chutes to a separate machine below, and these yield the final products—marketable zinc and iron concentrates. The whole plant treats 25 tons of ore per day (but could probably be worked to a larger capacity). The stuff before roasting carries 8–9 per cent. of zinc, 20 per cent. of iron, and the balance as combined sulphur and gangue. After the two separations the iron concentrate runs 3–5 per cent. of zinc, and the zinc concentrate carries 8–9 per cent. of iron and 36–44 per cent. of zinc.

The separator consists of a copper drum, about 2′ 6″ long, and of the same diameter; it is mounted on an insulated, horizontal axle and has a slow revolving motion. Within the drum, and occupying the sector of a quadrant, there are several long stationary electro-magnets, connected by insulated wire with a Gramme dynamo. The roasted ore moves in a thin layer down an inclined chute and strikes the cylindrical surface of the drum within the limits of the magnetic field, and where the surface is moving upward. The non-magnetic ore falls at

once into a receiver, but the magnetic particles adhere and are carried over the crown of the drum ; after passing beyond the magnetic field they drop into a separate receiver. The whole apparatus is enveloped by a hood, and all the dust is drawn from it by an exhaust fan. The process seems to work smoothly and give satisfaction.

A number of other electro-magnetic separators have been brought to public notice,[1] yet with the small experience thus far gained, it would be surprising if there did not still remain much room for improvement.[2] An active competitor with certain magnetic concentrations is sure to be encountered in the steadily-spreading wet processes for the metallurgical treatment of low grade copper ores.

FEATURES OF MILL CONSTRUCTION.

In no mill or factory does smooth and regular running depend more upon small, apparently unimportant, mechanical parts, than in an ore dressing establishment. Continuous automatic mechanism necessitates in very first instance the use of the soundest material and best workmanship, not only for the separating machines themselves, but for all the accessories —the shafting, belting, elevators, conveyors, etc. The mill structure, also, particularly if designed with several floors or stories, must be of very substantial construction to withstand the continual vibrations produced by crushers and jigs. Several cases could be cited of mills which, though excellent in general arrangement, have proved comparative failures, solely because in all their parts they were of too light a design. The newer mills comport in details of shafting, setting, replacing of wearing parts, etc., with present views of durability and economy; many of the clumsy and at once weak and inefficient mechanical designs and mill plants, familiar to the readers of Gaetschmann and Rittinger, have all but dropped out of sight in the steady march of improvement. The recent introduc-

[1] See, for example, the description of Vavin's Separator, which is said to extract 2 tons per day of magnetic sands or scales from their admixture with other bodies. *Revue Universelle*, 1881. Also the description of Edison's Separator, which appeared in the technical journals in 1880.

[2] As index to the path which investigation has been pursuing on this subject in America, the reader is referred to the article of Messrs. Eustis and Howe, "Contributions to the Metallurgy of Nickel and Copper." Trans. Am. Inst. of Mining Engineers, 1881-82, Vol. X., p. 305.

tion of iron framing for sets of sizing trommels[1] and the construction of iron battery frames, though approved by engineers of eminence, will hardly, at present, meet with a general indorsement. The increased cost of iron construction and the deterioration by rust and vibrations are points brought to bear against it.

In localities where the mills are run solely by wheels or turbines, the variability in the water supply sometimes throws a portion of the mill into idleness. To prevent, at such times, any interruption of mining operations, some dressing works are furnished with large storage bins which are fitted with chutes or automatic feeders allowing ore to be gradually drawn from them into the mill at a very small expenditure of labor. Many of the works, however, have permanent or auxiliary steam engines, and quite a number of these are of most approved patterns, and of high efficiency.

An excellent practice prevails of lining with sheet iron all launders but those conveying feed water. One or two mills have iron launders, but these are not to be generally recommended, Before setting a new plant in operation, all the separators, conduits, etc., are thoroughly soaked, to swell and tighten the joints, by allowing clear water to flow through them for fully three weeks. Work is then begun, and is characterized throughout by careful, intelligent supervision and economic management.

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[1] By the Humboldt Machine Shops at Kalk, near Cologne.